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CONTENTS

ONWARD SOVIET METALLURGY

The June Plenum of the Communist Party Central Committee has formulated some highly significant measures for the practical realization of the historical decisions of the 21st Party Congress concerning the most advanced technical progress in all branches of the national economy.

Soviet metallurgy is called upon to furnish high-grade metals for the growing demands of Soviet industry, transport, and agriculture. Increased metal production must be attained by widening the metallurgical base and by the most advanced technical improvement in apparatus and technical processes.

The figures for the first half of 1959 show that metallurgy has successfully fulfilled the tasks set for it. During this period pig-iron production was 21 million tons, steel, 29.3 million tons, and rolled steel, 23.1 million tons. 46.3 million tons of iron ore was extracted. In comparison with the first half of 1958 this represents an increase of 1,715,000 tons of pig iron, 2,345,000 tons of steel, and 1,670,000 tons of rolled ferrous metals.

There has been much technical progress in ferrous metallurgy. Recently, in the factories of the Sverdlovsk Council of National Economy agglomerate percentage in the blast furnace charge was increased from 63 to 82%. At the same time such measures as increasing gas furnace pressure, automation of scale cars, etc., allowed blast furnace production to rise 5.5% while coke consumption decreased 8%.

The workers of the "Azovstal" factory, in cooperation with a series of project, research and training organizations, have worked out a general scheme of complex blast-furnace automation, using computing-operating machinery.

A few factories of the Stalinsk Council of National Economy have completed their change-over to natural gas. This allows them to save about one million tons of coke per year.

The June Plenum disclosed the vast potential for increasing metal production that is at the disposal of our metallurgical factories.

The Plenum noted that delays in the construction of new mines, sintering and dressing plants and rolling mills are noticeably holding back the development of metallurgical industry.

Many Russsian Federation factories operate obsolete mills. Their productivity is 20-30 times lower than that of modern mills and they require a considerable amount of manual labor. Some of these mills must be replaced and some modernized. This will allow a significant increase in rolled-steel output without increasing the requisite space and will improve and relieve working conditions.

Sweeping measures must be taken in the next few years for the mechanization and automation of the productive processes in the metallurgical industry.

In ferrous and nonferrous metallurgy, automation is planned for 250 items in the mining industry, 114 in blast furnaces, 177 in open-hearth furnaces, 45 in rolling mills, etc.

The Seven-Year Plan for national economic development calls for an average of 80% mechanization and automation in production operations in ferrous metallurgy. The execution of basic measures along this line will cost 3.7 billion rubles, but this will pay for itself in about three years. 50,000 men will be freed during the Seven-Year Plan for work in other industrial branches.

In 1959-60 in the RSRSR, it is proposed to open three powerful sheet-rolling mills: mill 2800 at the Cherepovets Factory, mill 2800 at the Orski-Khalilovsk Combine, and mill 2500 at the Magnitogorsk Combine.

It is planned to expand significantly the Novolipets Factory to form the base for the development of a huge metallurgical combine in the center of European Russia that will rival Magnitogorsk. Its basic products are to be sheet metal, electrowelded pipe and also high-quality transformer and generator sheet for factories producing electrical equipment.

The decision has been made to carry on work at several ferrous metallurgy plants (among them the Magnitogorsk, Kuznets, and Nizhne-Tagil Metallurgical Combines toward complex mechanization and automation of the productive processes so as to develop them in the course of the next 4-5 years into experimental - demonstration plants whose experience can be put to use throughout the country. All this will allow a significant increase in productivity at these plants.

The Plenum emphasized that the struggle for technical progress in the national economy would be the decisive factor in the successful completion of the Seven-Year Plan.

According to the Plenum decisions on ferrous metallurgy the following should be practiced more widely: the refining of ores and agglomerates before melting, higher blast temperature and gas pressure in blast furnaces, use of natural gas and oxygen in blast furnaces and in steel melting, converters and electrometallurgy, the smelting of semikilled and low-alloy steel, continuous casting and vacuum treatment of steels, continuous speed rolling; the development of economical shapes in rolled steel, new high-grade steels and alloys, various new designs in pipe, standardized metal, metallic wire for pin manufacture, and heat-treatment of rolled steel, pipe, etc.

The Plenum resolutions on automation directed that basic consideration be concentrated on solving the problems of change-over from automation of separate

productive operations to its development for full technological processes, shops, and factories, primarily in those branches where automation will secure maximum economic advantage. As experience with automation in three mills at the First Ural New Pipe Factory and a rail-structural mill at the Nizhne-Tagil Metallurgical Combine has shown, productivity is increased 10%. The successful experience of the Sverdlovsk CNE plants with the automation of rolling mills must be quickly disseminated to all the metallurgical plants in the country.

The Soviet metallurgists have met the resolutions of the Party Penum with satisfaction and with yet greater strength have entered into socialist competition to reach the first year goals of the Seven-Year Plan ahead of schedule.

The great diligence of the metallurgists, enthusiastic over the historical resolutions of the Twenty-First Congress of the Communist Party of the Soviet Union, seems to guarantee that the tasks assigned them at the June Plenum will be successfully completed.

CHARGING EQUIPMENT WITH DOUBLE-WALL CONE

An article by I. L. Kordabnev appeared under this title on page 7 of the No. 3 issue of Metallurgist [page 97 of the translation]. Below are comments on this article received by the editors.

N. L. Ovcharenko

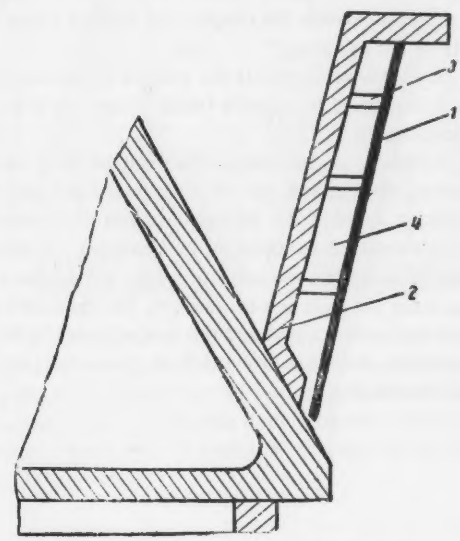
Dnepropetrovsk Metallurgical Institute

Many proposals for improving the charging mechanism have appeared lately. The majority of these attempt to deal with the abrasion damage to the bell and the cup contact surfaces. Such measures may result in somewhat increased life of the charging mechanism, but do not prevent leakage between the bell and the cup caused by warping of the charging-mechanism structure which results from frequent heating and cooling.

I. L. Kordabnev's proposal to construct the charging-mechanism cup with a double wall and to sweep between these walls partially cleaned gas which would prevent the dust-laden gas from being drawn into the interbell space may, with some improvements in construction, provide a far-reaching solution to the problem of sturdiness of the charging mechanism. With such a cup design the furnace gases will give up their heat to the outer cup wall. Heat transfer away from this wall and into the interbell space will be across a gas film whose heat transfer coefficient is very low (about 0.04 kcal / m·hr·°C) and consequently the temperature of the second wall will always be lower than that of the first.

Under steady operation the temperature of both cup walls will fluctuate less and this will prevent warping of the charging-mechanism structure. The seal between the bell and the cup will not deteriorate and surface abrasion will decrease.

If during steady-state operation the gas temperature in the furnace dome is 500°C, and the temperature in the interbell space is 100°C, then assuming that the outer and inner surfaces will heat up in proportion to these temperatures, the outer cup wall will thermally expand more than the inner. In our opinion a small clearance should be left between the rim of the outer wall and the bell (see sketch). This wall should be fabricated separately from the cup and should also be mounted separately.



Charging mechanism with a double-wall cup. 1) Outer cup wall; 2) inner cup wall; 3) strengthening ribs; 4) space filled with partially cleaned gas.

* * * *

L.A. Dem'yanets and D.A. Storozhik

Dnepropetrovsk Metallurgical Institute

Blast-furnace charging mechanisms invariably get out of order because of wear and tear on : 1) contact surfaces of the large bell and the cup; 2) contact surfaces of the small bell and the distributor funnel; 3) large bell stem in the lower bushing area of the small bell stem.

The method of increasing contact dependability between the large bell and the cup proposed by I. L. Kordabnev is, in our opinion, workable. However, in its implementation experience with the performance of annular groove seals into which vapor or gas is introduced should be considered.

Such seals proved unworkable because during the down cycle of the large bell the dust-laden gases whose pressure is 0.05-0.15 atm higher than that of the partially purified gas entered the supply pipes, plugged them and thereby put the seals completely out of commission. An analogous phenomenon is noticed in the gas lines of the equalizing valves. Therefore, the valve which supplies gas to the space between the walls should be closed while the large bell is in its down position.

I. L. Kordabnev does not mention in his paper how the decrease in volume of the interbell space due to the pockets between the walls of the cup was compensated for.

A double-walled cup is about one and one half times as heavy as one of conventional design This may necessitate mounting the bell and the cup separately in the furnace dome, because of insufficient load-bearing capacity of the girders.

The following claim made by the author raises some doubt: "Because there are two widely spaced contact surfaces, the bell cannot run out of true when being closed and therefore always fits snugly onto the cup along the whole of the periphery."

Calculations show that the amount of potential warpage is decreased by about a factor of two but is not eliminated entirely.

In order to protect the contact surface from abrasion by falling charge, and also to better brace the cup, the inclination angle of the bell at the point of contact with the cup should, in our opinion, be increased. This will further decrease the tendency to warp. For purposes of eliminating the gaps due to warpage, the spherical contact surface between the bell and the cup proposed by N. K. Borodenchik, A. I. Dikalov and D. A. Storozhik (see figure) should be tested.

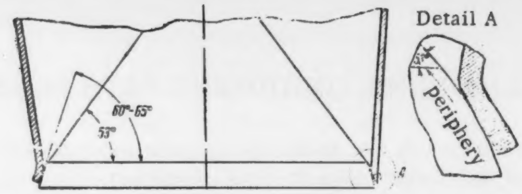


Fig. 1. Schematic diagram of a bell with spherical contact surface.

If the bell contact surface is harder than that of the cup -- this can be achieved by hard-surfacing of the bell then the high specific pressure that is exerted when the bell is closed will tend to deform the cup. The width of the contact surface will, according to our calculations, be about 5 to 7 mm. If the cup contact surface is also made spherical, which, mechanically, is very difficult, then the contact width could be made equal to that found in conventional charging mechanisms.

It is not possible to agree with I. K. Kordabnev that with satisfactory double-wall cup operation the small bell will break down solely because of abrasion by the charge. For example, at the Magnitogorsk Metallurgical Combine, where large bells have a working life of 2 to 3 years and where the pressure in the interbell space is not excessive (basic feature of the equalizing valves), the small bell wears out in approximately one year due to abrasion by dust-laden gases and also due to charge abrasion. The same behavior will prevail also in the double-wall cup charging mechanism.

The large bell stem will probably wear to the same degree as in the conventional mechanism when employing the equalizing-valve cycle.

RAPID REPAIR OF A BLAST FURNACE

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"Yuzhdomnaremont" (Southern Blast-Furnace Repair Group)
Construction Administration at Zhdanov.

During February and March, 1959, a blast furnace at the "Azovstal" Plant was given a general overhaul. Furnace repairs, together with some reconstruction, took 36 days and 20 hours, instead of the 45 days planned. This was accomplished as a result of the following fundamental measures of an engineering and organizational nature:

1) removal of all "salamander" in liquid form,

2) switching the furnace, just before shutdown, to smelting of fluid, high-phosphorus pig iron, which made possible the removal of "salamander" as a liquid,

3) mounting and welding of the crucible mantle simultaneously with furnace-bottom dismantling,

4) mechanized laying of carbon blocks,

5) simultaneous work at a number of levels,

6) well-timed provision of materials and tools on the site of the furnace to be repaired.

During the overhaul the lower part of the furnace was rebuilt; the thickness of the bottom was increased from 4140 mm to 5020 mm and foundation thickness was, at the same time, decreased by 1500 mm. The original bottom consisting of 12 courses of firebrick was replaced by a bottom with 2 courses of firebrick, 4 courses of carbon blocks and 7 courses of combination masonry (the periphery consisting of carbon blocks and the center of high-alumina bricks of large dimensions). The crucible was lined with 5 courses of carbon blocks to a height of 2 m (see Fig. 1).

The rivetted mantle around the crucible, tuyere zone, and bosh was replaced with a welded one; chill plates were installed, whereby one extra row of plates was added in the crucible; the fire-resistant masonry was renewed; waste gas burners, gas uptakes and a new charging mechanism were erected, and up to 20% of the inwall-mantle plate was replaced.

Because the inwall-support ring was lowered by 200 mm an additional piece of inwall mantle was installed. In addition the checkerwork, damstone wall, and dome were replaced in one of the stoves, and in the other two, combustion chambers were repaired and the checkerwork and lining partially renewed. The cleaned-gas duct was erected; the load-bearing capacity of the mounting girder was increased from 45 to 60 tons; rails on the skip bridge were changed; ore and coke bins and the skip pit were repaired, etc.

During the overhaul and reconstruction, 2300 cm^3 of fire-brick and 560 tons of carbon blocks were laid.

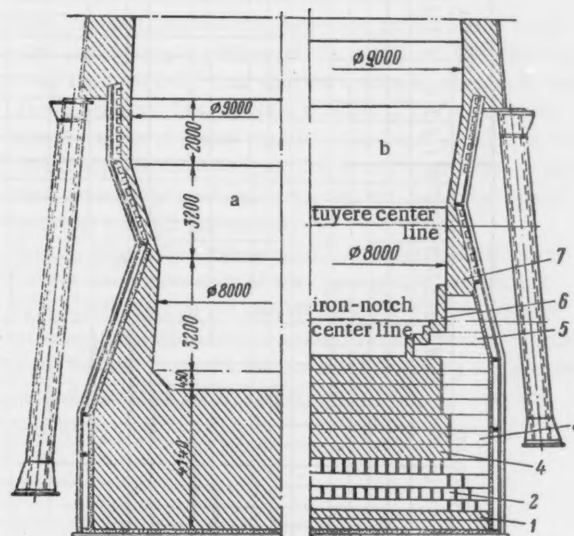

Fig. 1. Lower part of the blast furnace – a) before overhaul, b) after overhaul; 1) fire brick, 2) carbon blocks, 3) peripheral carbon blocks, 4) large-dimension high-alumina brick, 5) crucible carbon blocks, 6) protective lining of firebrick, 7) firebrick masonry in the crucible and tuyere zone.

The record of the work is represented graphically in Fig. 2.

In order to shorten the repair time, the "salamander" was drained in liquid form through specially constructed tapping holes. In order to obtain a sufficiently fluid pig iron, bog-limonite ore was included in the ore charge (60%) starting six days before shutting down the furnace for repairs; this resulted in pig iron which contained 0.77% silicon, 2.0% manganese, 0.06% sulfur, and 0.98% phosphorus. Four days before shutdown the content of bog-limonite aggregate was reduced to 35%; then pig iron contained 1.06% silicon, 2.16% manganese, 0.055% sulfur, and 0.84% phosphorus. The last 10 charges before shutdown consisted of coke alone.

Blowing-out the furnace began at 7 pm, February 4, and at 2 am, February 5, the iron notch was opened and 90 tons of pig iron tapped through it. After blowing out the furnace, 751 tons of pig iron was tapped through the

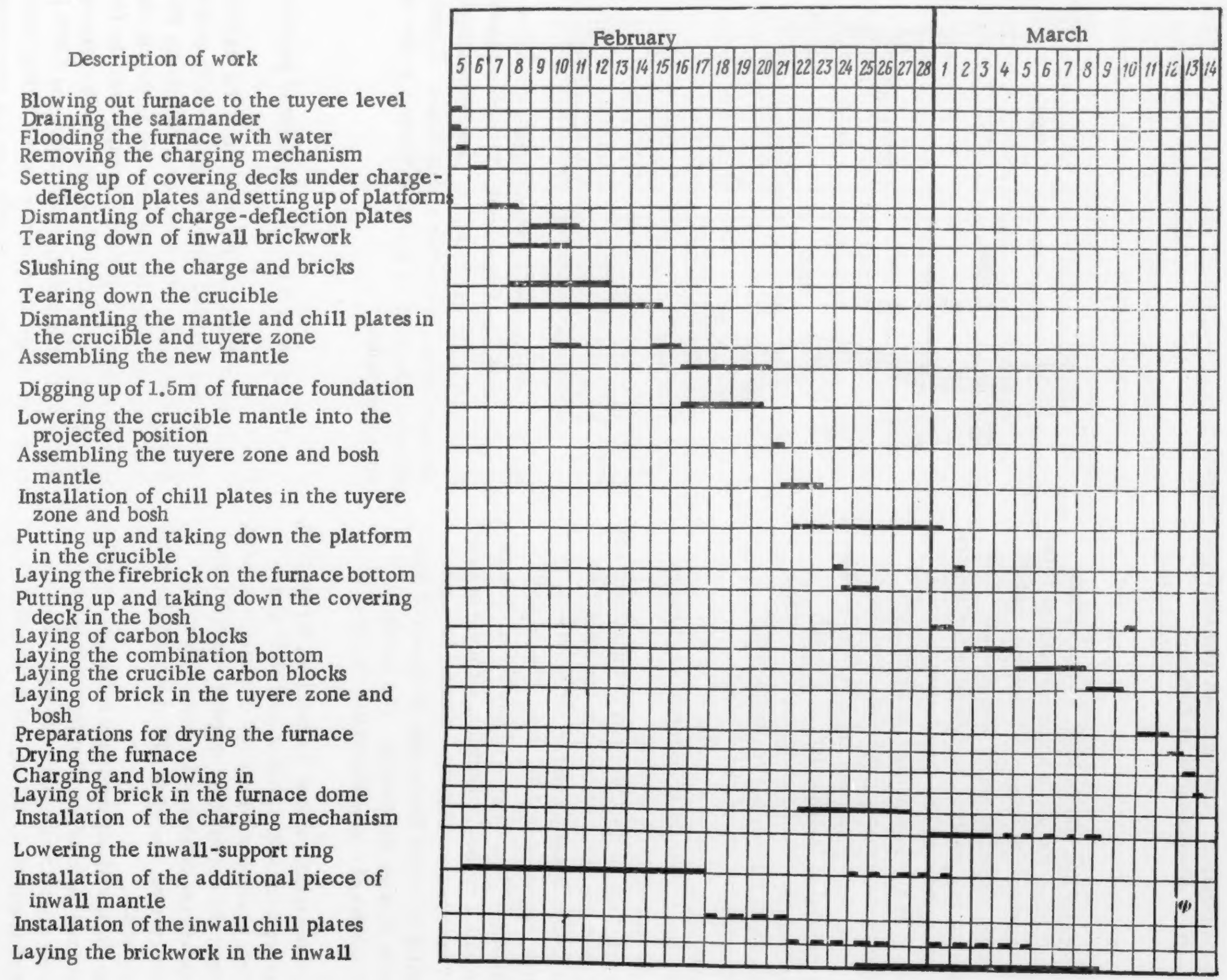

Fig. 2. Graph of the furnace-overhaul schedule.

upper salamander tap hole, which was located 2.5 m below the iron-notch center line. At the end of this tapping, pressure in the furnace was reduced to 0.3 and the blast rate was 1300 m^3/min at 750-800°C. As soon as slag appeared, the lower salamander hole, which was located 4.2 m below the iron notch (on the slag side), was burned open and 45 tons of pig iron was tapped into metal ladles, lined with fireclay brick, which were placed on the casting lorry.

After draining the salamander the blast was shut off, the furnace was flooded with water through four pipes installed beforehand in the furnace dome, the charging mechanism was dismantled and two metal platforms were suspended in the inwall by means of metallic cables. Masonry above the inwall chill plates was blasted out. A solid covering deck was erected below the charge deflection plates and from its main beam a tackle block for raising and lowering the platforms was suspended. One of the platforms was for work in the inwall and the other for lowering the inwall support ring.

The waste-gas burners were disconnected from the furnace and blind flanges were installed in their lower part.

Installation of platforms and covering decks enabled simultaneous work at five levels: crucible and bottom, inwall-support ring and lower inwall, charge-deflection plates and charging mechanism, gas uptakes and waste-gas burners.

In order to speed up demolition work in the crucible, four holes, 2700 by 2500 mm, located opposite each other, and also a hole 3500 by 2500 mm just above the slag notch were pierced; through these holes the crucible was cleared with the help of a "slusher." Thereafter the materials were loaded through special openings in the casting-pit ceiling and furnace-bottom-enclosure walls by means of tackle blocks and conveyors into pots and flatcars.

The replacement of the thin-walled bosh with a thick-walled one, the lowering of the inwall-support ring, and the modification of the upper portions of the support columns took place in four stages (see Fig. 3):

First stage. Cantilever brackets were put under the bustle pipe and the upper portions of the inwall-support columns were strengthened. This enabled cutting out the inner part of the columns (furnace side) in order to install the additional section of inwall mantle. The lower portions of the throat-support columns were also strengthened and, in addition, structural members were installed inside the inwall to insure strength and rigidity.

Second stage. The thin-walled bosh mantle was removed – resting the total weight of the upper part of the furnace (about 900 tons) on the support columns – and the bustle pipe was suspended from shortened suspension rods. In addition, the anchorage for the inwall-support ring was secured, whereby sections were cut out of the latter to clear the columns.

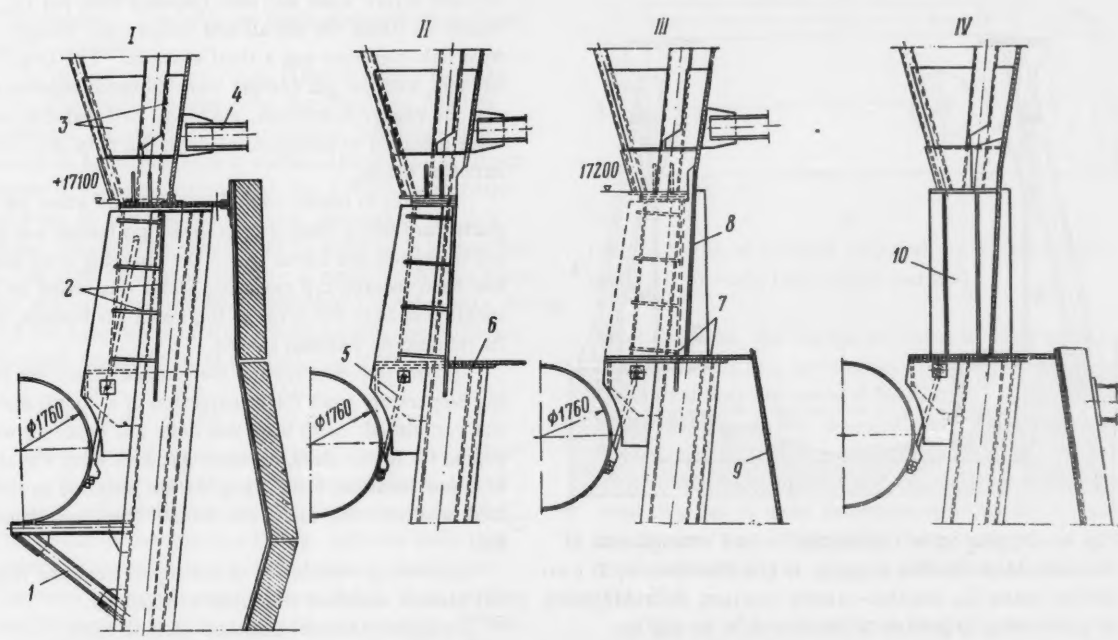

Fig. 3. Lowering the inwall-support ring (Roman numerals indicate stages of the work): 1) cantilever bracket under the bustle pipe, 2) strengthening of the upper portions of the inwall-support columns, 3) strengthening of the throat-column bottom, 4) strengthening structure, 5) shortened bustle-pipe suspension, 6) anchor supports for the inwall-support ring, 7) inwall-support ring components, 8) additional section of inwall mantle, 9) new bosh mantle, 10) permanent support pipe.

Third stage. The additional section of inwall and the new bosh mantle were erected. The bosh mantle and inwall-support ring were welded together.

Fourth stage. The upper portions of the old support columns, together with the strengthening members, were dismantled, a part of the inwall-support ring was welded, the inwall-strengthening structure was removed, and the lower portions of the support columns were secured.

After the inwall-support ring had been lowered, erection of the charge-deflection plates, chill plates, inwall, charging mechanism and other components began. The crucible mantle was secured 1.6 m above the foundation on cantilever beams welded to the main furnace columns; plates were lifted by means of hoists from the casting pit. Simultaneously with the assembly of the crucible the furnace foundation was broken up with pneumatic tools and light dynamiting (Fig. 4).

The crucible-mantle assembly which weighed 80 tons was placed in position to within 15 mm of the center of the inwall-support ring with the help of four 20-ton six-cable tackle blocks. An opening was left in the bosh mantle on the slag-notch side and through this opening chill plates were pulled in by tackle blocks.

After the chill plates in the bottom had been placed in position, the platform was raised for the installation of chill plates in the crucible; under its cover the concrete base was poured and two bottom courses were laid. Carbon blocks were placed after installation of the crucible tuyere zone, and bosh chill plates, and after mounting crossbeams in the bosh for suspension of a telpherage and a fully rotating beam. The carbon blocks – classified and packed in wooden crates (in accordance with masonry specifications) – were brought under the casting pit crane on flatcars, and unpacked there onto a wooden platform. Thereafter the blocks were hoisted onto a roller conveyor and pushed into the furnace through the slag notch. Seams between blocks were filled with a carbon paste, heated beforehand in blast stoves, followed by tamping with pneumatic rams. For thin seams the carbon paste was heated not only in blast stoves in which coke-oven gas was burned, but also in a tank with a steam coil. Thanks to such preheating of the paste and blocks, a high-quality masonry was insured. Following the laying of two peripheral courses of carbon blocks, building of the furnace-bottom center began; the bricks were supplied by conveyor through the iron notch.

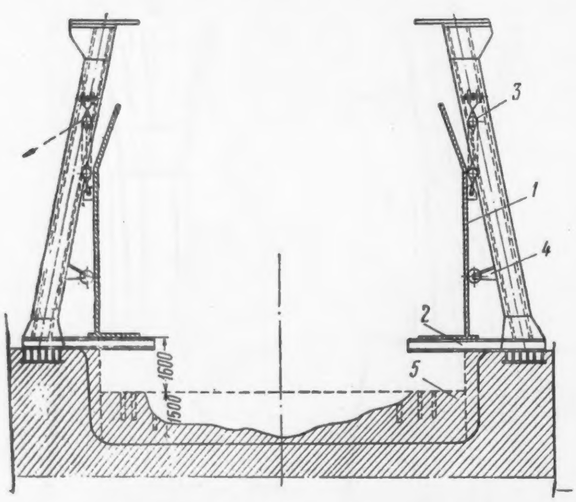

Fig. 4. Digging up of the foundation and arrangement of the assembled crucible mantle; 1) crucible mantle, 2) cantilever beams for crucible-mantle erection, 3) tackle block 4) guide rolls, 5) portion of foundation to be dug up.

In order to maintain comfortable temperatures at the work locations in the furnace, steam was passed into the bottom and crucible chill plates. After laying of the bottom had been completed the carbon blocks were ground with great care to a circular contour by means of grinders fitted with pobedit* knives, carefully shielded, and 5 courses of blocks were laid in the crucible. Thereafter the cross beams in the bosh were dismantled and the monorail and rotating beam were taken down; the platform was lowered into the crucible and – working from it – the tuyere zone and bosh masonry were put in. Bricks for lining the inwall and throat were brought up by a belt conveyor and a shelf elevator. The large bell, the cup, and the gas damper were mounted separately, and the charge distributor, the small bell and the stems were mounted in common, all with the help of a construction crane.

In order to insure normal temperature when the blast ducts were being lined, a coke-oven gas burner was placed under each of the ducts. After the charging mechanism had been secured, the mounting girder was raised on two tackle blocks to the level of the waste gas burners, and its reinforcing was then started.

Apart from overhauling the furnace proper the following preparatory work for construction of a fourth stove was completed: earth work was done and concrete was poured for foundations, cleaned-gas duct work was shifted to a new location, blind flanges were installed in the cold and hot-blast ducts, the exhaust flues were lined, etc.

Following completion of the repair work the furnace was cleared and then was dried with hot air.

* A tungsten carbide–titanium carbide alloy.

INCREASING THE DURABILITY OF HAMMERS IN HAMMER MILLS

Cand. Tech. Sci. M. A. Tylkin
Dzerzhinskii Factory

At sintering plants which process fused sinter, the limestone is ground in hammer mills. The coarse limestone entering the mill is reduced by friction with the beater (hammers) and the fine fraction passes through the grate into a special bunker. The material is then screened, the 0-3 mm fraction going to the charge and the remainder being returned to the mill.

During operation, the beaters are subjected to abrasive action from contact with the limestone, and dynamic stresses from impacts with pieces of material. In most plants, the beaters are built up manually using T-590 and T-620 electrodes, or electrodes with a coating of stalinite. This building-up is very difficult and time-consuming. With manual building-up it is difficult to arrange that the weight of the restored components is the same, and the difference in the weight of the hammers leads to considerable imbalance in the rotor. Furthermore, manual building-up does not provide sufficient durability of the beaters. The built-up layer often becomes pitted during operation.

In recent years, the Paton Electrowelding Institute of the Academy of Sciences of the Ukrainian SSR has developed a method for the automatic building-up of beaters in an atmosphere of carbon dioxide, using direct current of reverse polarity with the A-537M automatic machine, by the method of forced forming of the new metal. The section which is to be built up is surrounded on four sides by a copper form – a mold, cooled by water. The material used was PP-U45Kh23G6T powder wire, fusing with the base metal of steel 40-45 to give the following chemical composition of the built-up layer, %:

C	Cr	Mn	Ti
4.2-4.7	21-23	3.5-5.0	0.4-0.6

Before building-up, each blank was heated to 550-600°. The built-up beaters were cooled slowly in the furnace. After building-up, the hardness was 54-57 R_C. The results of the Electrowelding Institute show that the life of experimental beaters built up by the gas-electrical method is several times higher when grinding coal (compared with beaters built up manually by electrodes with a stalinite coating). Owing to the fact that this method has not yet been extended to steel plants, it is not yet possible to judge the durability of beaters used for grinding flux.

In steel plants, extensive use is made of heat treatment to strengthen the beaters. Thus, at the Magnitogorsk

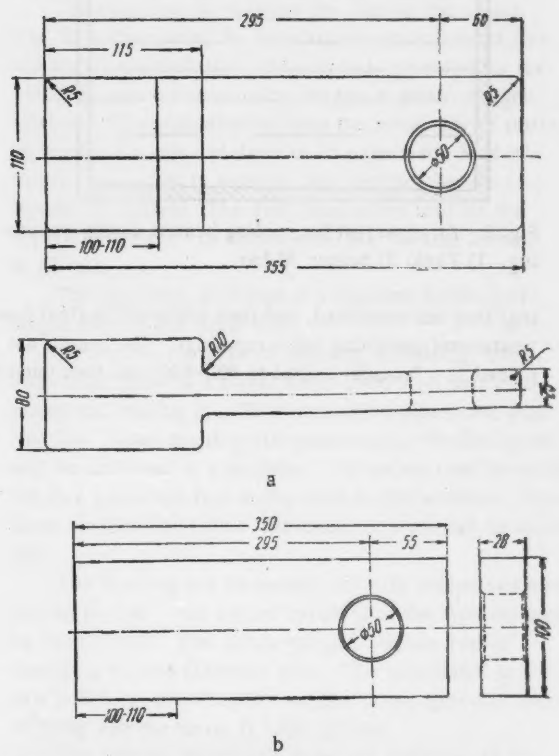

Fig. 1. Beater of hammer mill: a) made from forged material; b) made from rolled material.

Steel Combine, the beaters are made of 40Kh steel with heat treatment to a hardness of 48-51 R_C, at the Makeev Steel plant they are made of 50G2 steel to a hardness of 387-415 H_B, at the "Zaporozhstal'" Plant they are of 85 steel and treated to a hardness of 56-58 R_C.

At the Dzerzhinskii Plant two hammer mills have been installed to grind limestone, each having a capacity of 150 tons/hour. Two types of beaters are used in these mills: of forged material and of rolled material (Fig. 1).

The use of forged material for the beaters is perfectly satisfactory, despite the relatively high cost in comparison with beaters made from rolled material, since the forged beaters permit finer grinding of the limestone.

The forged and rolled beaters are made from 50G2 steel. After preliminary heat-treatment (usually anneal-

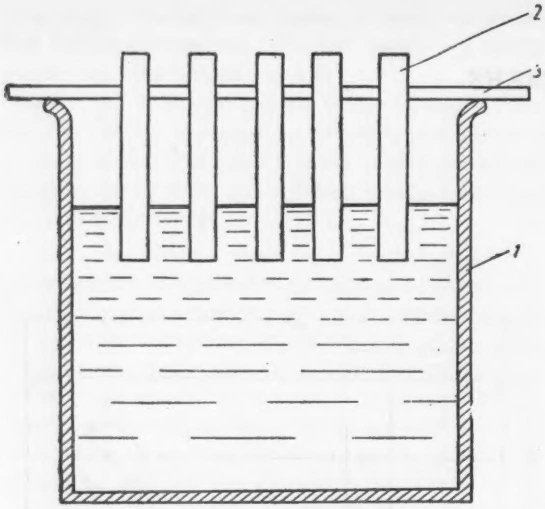

Fig. 2. Arrangement for cooling beaters during quenching. 1) Tank; 2) beater; 3) bar.

ing) they are machined, and then subjected to final heat-treatment (quenching and tempering). The beaters are placed in a furnace heated to 820-840° and then quenched in water. The working parts of the beaters are immersed to a depth of 100-110 mm in a water tank (Fig. 2), where they hang on bars placed along the length of the quenching tank, the level of water in which is varied to suit the required depth of quenching.

The beaters are soaked in water till the shank cools to 600° (beaters made of forged material are cooled for 6 min, beaters from rolled material for 4 min). To prevent the formation of a steam pocket around the beaters, they are agitated by rotating the bars. After quenching, the beaters are cooled in air, and then tempered by heating to 340-360° and soaking for 2.5 hours.

After heat-treatment, the hardness of the working part of the beaters is 415-460 H_B, and the shank is 196-241 H_B. The operating life of the machined forged beaters reaches 30 days, and those made from rolled material 15-20 days.

A drawback of this method is the fact that the beaters cannot be restored.

We are of the opinion that the most efficient method for strengthening beaters of hammer mills is by heat-treatment; worn beaters are best restored by automatic building-up using the gas-electrical method developed by the Paton Electrowelding Institute.

APPARATUS FOR TREATING PIG IRON IN THE MOLDS

A. A. Garkusha and A. Z. Ryzhavskii

Giprostal'

The preliminary blasting of pig iron with oxygen reduces the content of silicon and manganese and increases the temperature of the iron. When this iron is used in the open-hearth process, the consumption of ore and limestone in the charge is considerably reduced, there is a corresponding decrease in the quantity of slag which, together with the increased temperature of the iron, means that the smelting time can be reduced.

Studies at the Ukrainian Metals Institute showed that the introduction of oxidizing and slag-forming additions to the ladle cause an increase in the loss of impurities in the iron during the filling and transporting of the ladle and an increase in the loss of silicon during blasting with a steam-oxygen mixture.* Of the additions tested, the best results were obtained by adding 15 kg or ore and limestone per ton of iron.

Based on these studies, Giprostal' has developed an arrangement for the preliminary treatment of iron in the ladles the blasting taking 30 min; in this arrangement the iron is desulfurized in the ladle, and the steam-oxygen mixture is used without flue gas scrubbing (the steam consumption is 6 kg/ton of iron, the pressure of the steam 7-10 atm). The planned oxygen consumption for the blasting of one ladle is 800-2000 mm^3/hour, the pressure 7-10 atm, the consumption of oxygen per ton of iron up to 7 mm^3 (NTP).

The installations for making additions to the pig-iron ladles (Fig. 1) have been planned for the blast-furnace shop. The usual type of ore is used – coarse open-hearth ore, the limestone (15-25 mm fractions) is supplied by the sintering plant. For reasons of safety, the additions are made to the stream of iron while the ladle is being filled. To prevent splashing during blasting, not more than 65-70 tons of iron should be poured into a 100-ton ladle; for this purpose, instruments are provided to indicate the iron level.

The additions are made from bunkers with feed mechanisms, arranged over each iron-carrying ladle. The materials are fed to the bunker by a casting-yard crane from boxes (containers) of known capacity, the volumes of which are 0.45 m^3 for the ore, and 0.65 m^3 for the limestone, corresponding to a dry weight of about one ton. While the ladle is being filled with iron, the additions are gradually made (over 3-5 min) to the ladle by means of a tray-type feeding mechanism.

Since the ladles will not be completely filled, the time for their circulation increases and there is a certain reduction in the life of the lining, the number of operating ladles should be increased by approximately 1.5 times.

The apparatus for treating the iron in the ladles (Fig. 2) is calculated for simultaneous treatment of five ladles. The installations are primarily intended for the desiliconization of iron using oxygen or steam-oxygen mixture. The plan also includes the possibility of setting up service bunkers and devices for pneumatic feed of lime to the ladles in order to desulfurize the iron (a secondary feature). The daily production rate for the apparatus is up to 120 ladles when the blasting lasts up to 30 min.

The apparatus is placed in a separate building of length 42.4 m and width 6 m.

To remove the gaseous products of blasting, over each ladle there is a metal hood, lined with cast iron plates and leading to a 22 m extraction pipe lined with fireclay. When blasting with pure oxygen, the flue gases will be delivered to a scrubber. Valves are used to switch the flue gas to the flue or the lead to the scrubber. Several areas are provided where maintenance work can be carried out.

The blasting can be carried out with cooled and non-cooled tuyeres. The cooled tuyere is of the type designed by TsNIIChMt. The steam-oxygen mixture leaves through a 45 mm diameter hole. The calculated speed of exit is 300 m/sec when the oxygen consumption is 2000 m^3/hour and the steam is 1500 kg/hour.

The oxygen, steam and water are delivered to the tuyere by means of a flexible hose, connected with the tuyere by means of connecting tubes with union nuts. The length of the cooled tuyere is 7300 mm, the diameter 114 mm.

The noncooled tuyere consists of a mixer and an interchangeable part. Oxygen and steam are delivered to the mixer, as in the case of the cooled tuyere. The mixers are connected with the interchangeable part by means of nuts. The length of the mixer is 4000 mm. The maximum length of the interchangeable part is 6200 mm, diameter 48 mm.

When working with pure oxygen, the exit gases together with air at a temperature of about 700° enter the checkerless scrubber with 3-stage annular pipes with involute nozzles, providing a fine spray of water to cool and clean the gases. From the scrubber, the gases enter

* Metallurg 8, (1958).

† (Central Scientific Research Institute of Ferrous Metallurgy).

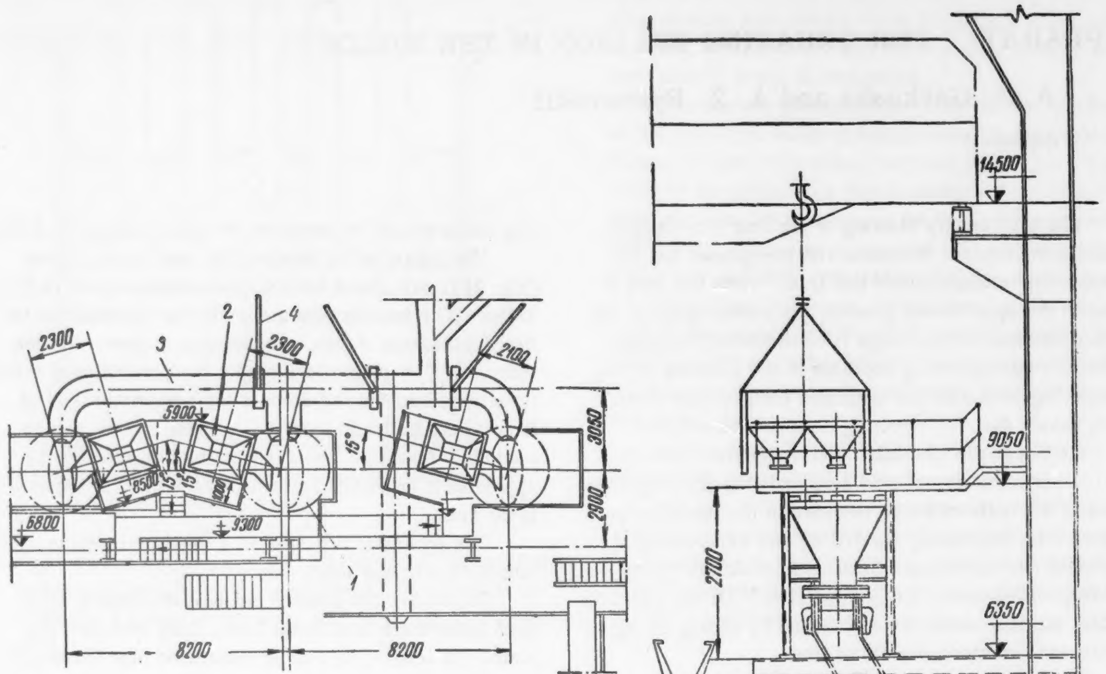

Fig. 1. Installation for feeding additions to pig-iron ladles: 1) ladle; 2) bunker for additions; 3) chute; 4) cut-off paddle.

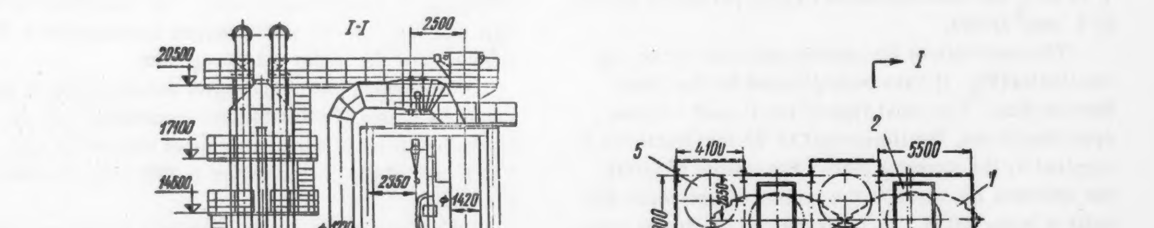

Fig. 2. Apparatus for blasting iron. 1) Ladle; 2) extraction hood; 3) scrubber; 4) flue-gas collector; 5) site for bunker with lime for desulfurization; 6) control station; 7) flue-gas extractors.

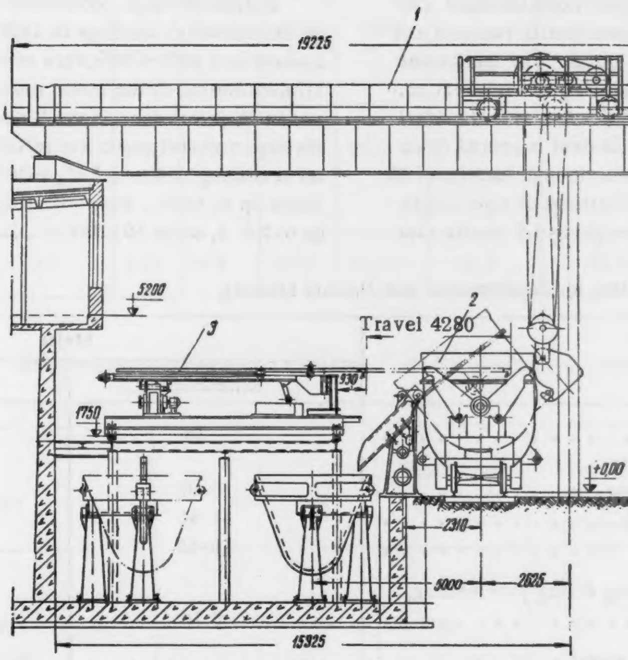

Fig. 3. Section for running-off slag: 1) Crane; 2) ladle; 3) machine for running-off slag.

two spray tubes, in the reduced section of which there are water sprays. In the spray pipes, the gases are finally freed of dust, and on leaving these pipes, the gases pass through a water trap, which removes the entrained water and then the flue-gas extractor puts the gases into the atmosphere through the flue pipe which is common to all five installations. The calculated quantity of flue gas is 4500 m^3/hour and, allowing for the air, there is a total of 1800 m^3/hour for one ladle.

This arrangement for gas purification does not allow for recovery of the heat of the exit gases, since the efficiency of the recovery boilers would be low due to the intermittent operation of the equipment.

Bunkers are used for desulfurization of the iron with lime. The 0-1 mm lime fraction will enter from a grinding section which is specially provided for this purpose in the VNIIPTMASh‡ type reservoirs, equipped with devices for pneumatic unloading. Nitrogen was chosen to carry the lime into the iron although this does not exclude the possibility of using other gases. The nitrogen will be supplied from the oxygen station to a compressor in the installation where it will be compressed to the desired pressure.

Since at the present time there is no reliable information on the order of desiliconization and desulfurization operations and on the combination of them, the desulfurization of the iron has been assigned to the second stage in the construction of the installations.

The apparatus for running off the slag after blasting (Fig. 3) is situated in front of the mixer section. It is designed for the simultaneous running off of slag from three ladles. After the hot-metal cars have been installed, the ladles are tilted by means of a 75-ton crane trolley and the slag is run off by a special machine into a slag tray located in the pit. The tray is changed by the same crane trolley by means of a cross arm. The tray is tilted, the slag run off and the tray changed by remote control.

Calculations showed that the installation pays for itself in two years due to the increase in productivity of the open-hearth furnaces and the reduction in cost of the steel.

‡ (All-Union Scientific Research Institute of Hoisting and Conveying Machines).

THE MELTING OF STEEL Zkp WITH LOW MANGANESE CONTENT

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Kuznets Metallurgical Combine

At the Kuznets Metallurgical Combine, steel Zkp is melted in large-capacity, open-hearth furnaces and poured into two steel-pouring ladles. The manganese content in steel Zkp varies according to technical conditions between 0.40% and 0.60%. As a rule the steel is deoxidized in the ladles. The steel is poured down from the double-stoppered ladles through buckets of 35 mm diameter into continuous castings of right-angle cross section, forming ingots weighing 6.9 metric tons.

In attempting to economize on ferromanganese, 30 experimental meltings in 1958, with a manganese content of 0.25%-0.35%, were carried out. In 18 melts all deoxidation of the metal took place in the ladle and in 12 it was done in the furnace. The metal from the experimental melts was rolled out in small shapes set according to Group "A", GOST 500-52 and 380-50: sheets up to 6mm, discs with diameter 25 mm, angles up to No. 4, strips 50 x 10mm, lining, facings, and roofing

TABLE 1. Basic Data for Experimental and Normal Melting

Characteristics	Melts	
	Experimental	Normal
Number of melts	18	33
Time, hours- minutes		
boiling, without ore	0-32	0-34
finishing	1-40	1-36
total melting	10-45	10-50
Rate of carbon burning during pure boiling, % C/hour	0.18	0.21
Temperature of the metal before pouring, °C	1585-1590	1585-1590
Content at melting out, %		
C	0.69	0.72
P	0.031	0.032
S	0.045	0.043
Content before deoxidization, %		
C	0.17	0.16
Mn	0.04	0.05
P	0.018	0.017
S	0.035	0.036
Ferromanganese added for deoxidation (average), kg	1755	2612
Diameter of steel-pouring buckets, mm	35	35
Ingot-pouring time, weight 6.9 metric tons, minutes	2-3.7	2-3.7
Boiling time of metal in castings, minutes	12-15	11-14
Composition of finished steel, %		
C	0.18	0.17
Mn	0.32	0.45
P	0.023	0.022
S	0.033	0.030

TABLE 2. Surface Quality of the Metal from Experimental and Normal Melts and Expenditure of Ferromanganese

Melts	Number of Melts	Surface quality, marks				Number of defective ingots, %		Amount of ferromanganese, kg/ metric ton steel	Average manganese content in finished steel, %
		average	distribution of melts according to marks, %			with flaws	with cracks		
			Mark 1	Mark 2	Mark 3				
Experimental	18	2.20	7.0	68.0	25.0	13.0	14.0	4.50	0.32
Normal	33	2.10	17.2	57.4	25.4	12.5	18.0	6.70	0.45

TABLE 3. Mechanical Characteristics of Rolled Steel Zkp from Experimental and Normal Melts

Shape	Melt	Yield Point kg/mm²	Yield Strength kg/mm²	Relative lengthening %
		Not less than		
Acc. to GOST 500-52, 380-50		24	38	25
Sheet, 4; 5; 6mm	Experimental	26.9	42.7	33.5
	Normal	27.2	40.7	32.2
Disc, 10; 12; 25 mm	Experimental	29.6	41.6	32.2
	Normal	28.8	42.0	33.8
Angle, 40 x 40 x 4mm	Experimental	30.2	42.4	32.7
	Normal	31.5	43.6	30.6
Strip, 5 x 10mm	Experimental	28.8	44.5	35.0
	Normal	30.4	43.9	34.2

sheets. Part of the metal was rolled out with varying shapes: lining - sheet, disc-sheet, angle sheet, etc.

For technological comparison of the melting, casting, ferromanganese expenditure and steel quality, data were taken from 33 normal melts of steel Zkp during the same period. The metal from these melts was also fully deoxidized in the ladle and manganese content in the finished steel was 0.40-0.60%.

The technology of the melting and casting under experimental and normal conditions was practically equivalent (Table 1).

When the necessary carbon, phosphorus, and sulfur content was obtained, the metal was poured from the furnace into a steel-pouring ladle. There was no deoxidation in the furnace. The last chemical analyses of metal and slag were taken from the furnace as pouring commenced. The furnace temperature of the metal was controlled during the melt by an immersed thermocouple.

Ferromanganese in pieces up to 50mm was introduced into the ladle at a steady rate from a stationary hopper before the slag began to form. The passage of the metal into the ladle lasting from the time that the melted steel was all out of the furnace to the commencement of pouring occupied about 10 minutes. In the experimental melts the metal boiled longer and somewhat more actively in the castings.

As shown by the values for metal surface quality obtained at the blooming mill (Table 2) the metal from

the experimental melts was superior to that of the normal.

Ferromanganese expenditure was only measured in 18 experimental melts and in 33 normal melts, in all of which deoxidation took place wholly in the ladle. As is obvious from Table 2, the experimental melts represent a saving of 2.2 kg ferromanganese/metric ton of steel over the normal melts.

The mechanical qualities of the rolled metal produced by the experimental and normal melts are shown in Table 3 The steel from the melts, rolled into lining strips, were tested on a bend of 60°; the results for all types of melts were satisfactory. Steel Zkp with low manganese content, rolled out to small shapes (sheet, strip, disc, angle, linings, etc.), has mechanical characteristics equivalent to those of normally melted steel Zkp and exceeds significantly the requirements for GOST 500-52 and 380-50.

On the basis of the foregoing the following conclusions can be made:

The melting, pouring, and rolling of steel Zkp with manganese content lowered to 0.25-0.35% involves no technological difficulties. The surface quality and mechanical characteristics of the rolled steel in small shapes satisfies GOST requirements. 2.20 kg ferromanganese per metric ton of steel are saved during deoxidation.

The Kuznets Metallurgical Combine has been producing steel Zkp with low manganese content for rolling into small shapes since August, 1958.

PERFORMANCE OF ARC - FURNACE ROOF COOLERS

A. F. Kablukovskii, S. D. Skorokhod and S. A. Zhikharevich

Élektrostal' Factory

Cooler installations in the roofs of electric arc-furnaces provide reinforced openings for the electrodes and cool the central part of the roof refractory which is subject to the most severe wear. Coolers usually consist of hollow, water-cooled annular rings or 25-45 mm diameter tubes. Earlier they were mounted above the roof, but it proved more rational to mount them within the roof refractory where they bring about better cooling of the central part and form a brace for the bricks around the electrode opening.

The arrangement of the coolers within the roof refractory of the Electrostal' Factory arc furnaces is shown in Fig. 1.

Intensification of the melting process — which came with the use of gaseous oxygen – and the increase in furnace-transformer capacity, replacement of Dinas brick in the roof with the highly refractory chrome-magnesite brick, and simultaneous increase in roof sturdiness, decreased the durability of the coolers 3 to 5 fold.

Changing the coolers during furnace operation is a very difficult and not always safe operation, and on a number of occasions has resulted in the most critical part of the roof being loosened and furnace life being shortened. Changing the coolers leads to furnace down time, decrease in furnace productivity, and reallocation of the metal to less critical orders (in cases where water gets into the metal bath).

Experience has shown that any influence of the roof refractory on cooler burn-through was related to the rate of spalling of the bricks and their ability to absorb iron oxides from the charge dust. A significant role is played here by the electrical conductivity of the roof brickwork at high temperatures.

High-alumina agglomerates, which are characterized by high purity, have better electrical insulating properties.

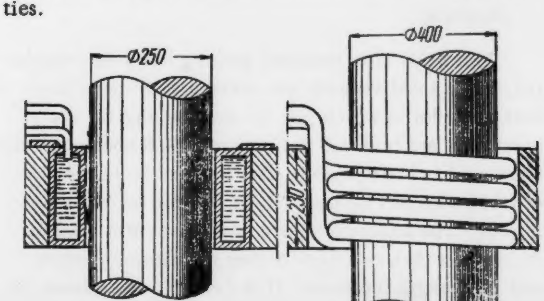


Fig. 1. Layout of coolers in the arc-furnace roof masonry.

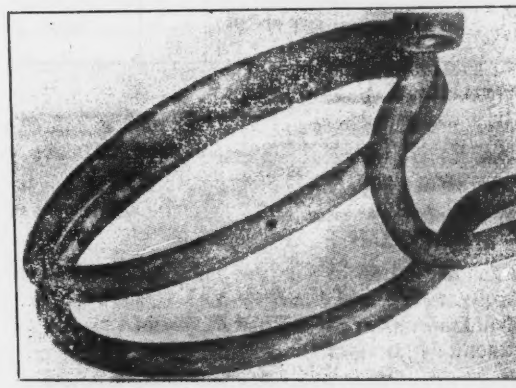


Fig. 2. Cooler burned through by an electric discharge resulting from a short circuit.

Because preferential diffusion can occur in the chrome-magnesite brick matrix, a laminar structure forms in the brickwork during service. Thus, the working layer (30-40 mm) is saturated with iron oxides and its electrical conductivity increases.

Chrome-magnesite roof brickwork is worn away chiefly through successive spalling of the working zone in layers of 30-40 mm every 10-20 heats. During this period a large amount of charge dust deposits on the working surfaces. In heats using oxygen the charge dust depositing on the brickwork contains up to 60% iron oxide and is a good electrical conductor even at room temperature. Under certain conditions, when the spalling of the working zone does not occur between two successive phases (10-20 heats) and the continuous conductive path formed by the charge-dust-saturated brick edges is preserved, short circuits appear during which the cooler walls burn through (see Fig. 2) both on the roof—refractory side and on the electrode side. During heats using oxygen, as many as 20 cases of cooler failure occurred during a single roof campaign.

In order to prevent cooler burn-through, numerous methods of insulating the coolers from the chrome-magnesite roof masonry, changing the design of the coolers themselves, etc., were tried at the plant.

Numerous experiments with insulating the coolers have shown that it is possible to avoid cooler failure by burning only when the coolers are fully isolated from the influence of the melt products in the furnace melting zone. Use of specially shaped, high-alumina brick for this purpose does not guarantee the tightness of seal

Basic Characteristics of the High-Alumina Concrete Components

Material	Percentage			High-temp. strength °C
	SiO_2	Al_2O_3	CaO	
High-alumina cement	—	72—75	23—25	1660—1710
High-alumina fire clay.	9	85	4—5	1850

and the structural sturdiness that is essential. Consequently special high-alumina electroinsulating concrete is being used at the plant in order to fully solve the problem of protecting the cooler from the action of the charge dusts.

The technology of preparing the high-alumina concrete has been worked out at the Ukrainian Refractory Research Institute * and is based on the use of a special cement and high-alumina fire clay. The basic engineering and technical characteristics of the concrete are given in the table. Maximum particle size of the high-alumina fire clay used in the concrete is 3 mm, including 25-30% of particles less than 0.09 mm.

The electroinsulating concrete consists of 20% high-alumina cement and 80% of the high-alumina fire clay fraction less than 3 mm. The particularly noteworthy characteristic of this high-alumina concrete is its rapid increase in strength on hardening and its retention of high strength on heating to elevated temperatures.

Batches of the high-alumina cement and high-alumina fire clay for the formation of the concrete mix were prepared at the experimental plant of the UNIIO (Ukrainian Refractory Research Institute). Experiments concerning electrical insulation of coolers by means of the concrete mix were conducted along two lines: 1) use of high-alumina brick between the basic chrome-magnesite masonry and the concrete, 2) direct contact between the concrete and the chrome-magnesite roof brick. The experiments showed that in both cases the results were the same and that complete elimination of cooler failures from burning is possible.

The cooler diameter is made 80 mm larger in order to accommodate the protective layer of concrete (40 mm for each side of the electrode). A square opening is left in the chrome- magnesite roof masonry for each cooler, allowing 50-80 mm clearance between the outer cooler periphery and the walls of the opening. In order to decrease concrete consumption, the corners of the opening are filled with chrome-magnesite brick. Metallic forms which provide 25 mm clearance on each side of the electrode are inserted into the square opening. A thin iron sheet covered with lubricating oil is inserted around the form.

The concrete mix is prepared on a clean platform in the roof-assembly area. High-alumina fire clay and cement, in the proper proportions and in a quantity sufficient for casting one cooler, are thoroughly mixed in the dry state and then water is added until a paste-like mass is obtained. In order to avoid absorption of moisture from the concrete by the brickwork in contact with it, the latter is saturated before pouring the concrete.

It is necessary to thoroughly mix the fluid mass before pouring in order to obtain the required uniformity and prevent settling of the clay. Crushed and moistened high-alumina fire clay (5-30 mm particle size) is added to the fluid concrete when filling the space between the roof brickwork and the form in order to impart to the concrete the necessary heat-resisting properties. This crushed brick is tamped down into the concrete with the help of wooden poles to pack it as densely as possible.

Concrete is poured to a level of 100-120 mm and thereafter the cooler is inserted into the clearance space. When the cooler has been placed and secured with wedges in the required position, pouring of concrete is resumed with addition of crushed brick and uninterrupted tamping. In order to cover the cooler tubes which emerge onto the roof surface, and to construct a horizontal plane for mounting a cast-iron "sealing dish", a special concrete section is formed which protrudes 70-90 mm (see Fig. 3).

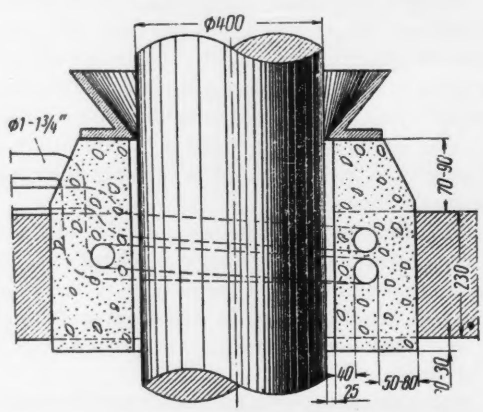


Fig. 3. Cooler insulation made of high-alumina concrete.

Two hours after concrete pouring has been completed, all exposed surfaces are covered with moist sawdust. The forms which provide the opening for the electrodes are removed not earlier than 8 hours after the concrete has been poured. The inner surfaces are also covered with wet sawdust after the forms have been removed. The concrete is held covered by wet sawdust for not less than two days; during this time it hardens and its strength increases. It is desirable to increase the

* In 1947 by Candidate for Dr. Eng. A. I. Roizen.

setting period to 3-4 days. It is not recommended to move the roof from one place to another during this time.

After being held in a moist atmosphere the sawdust is removed and the concrete is held for a day at room temperature. During the next two shifts the concrete is subjected to more intensified drying and warming-up; coke "salamanders" are placed into the electrode opening for this purpose. Final concrete-preheating temperature is 200-250°. During the cold seasons when the temperature in the roof-assembly area is low, hot water (50-60°) should be used for concrete make-up, and the poured concrete should be covered with old cloths or matting over the moist sawdust.

It is not necessary to heat the roof before erection on the furnace.

Material requirements for electroinsulating a coil-type cooler in the roof of a 15-ton electric furnace were as follows: 50-70 kg of cement, 250 kg of the 0.3 mm fraction of fire clay, and 250 kg of the 5-40 mm fraction.

Fifteen roofs of a furnace, in which 80-90% of the heats are smelted with oxygen blowing, have been insulated in the manner described above. There have been no cooler failures on these roofs. This has resulted in an increase in furnace productivity, elimination of metal spoilage resulting from water falling into the metal, and elimination of the labor-consuming changing of burned coolers.

Roofs of average durability, equipped with high-alumina concrete insulated coolers, withstood 61 heats without cooler failure as opposed to 57 heats for roofs without the insulation, in which the coolers failed six times during the campaign of each roof.

A 100 TON CONVERTER FOR BLASTING STEEL WITH OXYGEN

A. I. Pavlov and A. A. Perimov

Stal'proekt

The extensive use of oxygen in steelmaking has recently led to the development of a new improved method for converting open-hearth iron in converters by blasting with oxygen from the top. As well as having certain disadvantages in comparison with the open-hearth process (reduced yield and the necessity for gas scrubbing), the oxygen converter process has a number of advantages:

1) high productivity of the unit;

2) low capital costs for shop construction;

3) the possibility of using the heat of the waste gases, which covers the cost of producing the oxygen;

4) the low consumption of refractories (7-10 kg/ton).

Both in the Soviet Union and abroad, large converter shops are already operating. Along with the introduction of new converter shops, a continuous increase can be observed in the capacity of units with top blasting. Whereas the increase in capacity of basic (up to 60 tons) and Bessemer converters (up to 40 tons) has been very slow, during the last three years the capacity of oxygen converters has increased up to 70 tons. Operating experience has shown the possibility of using the existing converter with a double charge without serious deterioration in the process and without prolonging the smelting time.

The increase in the converter tonnage is accompanied by a decrease in the capital costs for building the plant, the manufacture of equipment, and operation; there is also an increase in labor productivity.

The "Stal'proekt" Institute carried out tests on a 100 ton converter with oxygen blast from the top and on the auxiliary equipment associated with it (hot metal car, jack car, etc.). The nominal tonnage of the converter was determined from the specifications of the system for hoisting the iron to the converter by a standard 100 ton iron-carrying ladle (Fig. 1).

The 100-ton converters are installed in a shop (Gipromez project) having the following basic dimensions (meters):

Length	216
Width	69
Width of converter bay	15
Width of casting bay	18
Width of pouring bay	18
Spacing in the columns of the row of converters	24
Height of working platform	8.15

The shop consists of mixer and converter sections and two casting bays. From the mixer section, the molten iron is taken in 100 ton ladles by electric car to the converter section and by means of a 125/30 ton casting crane it is poured into the converter. Buckets are used to charge the solid metal, hoppers are used to feed in the solid fluxes, the bunkers being placed over the converter and having chutes from them. The oxygen for the blast is delivered through a water-cooled tuyere.

The steel is run into a steel-casting ladle, mounted on a powered car, which moves transversely under the converter at the shop floor level. The slag from the converter is poured into a 16 m^3 slag container. In order to pour the iron and charge the scrap, the converter is tilted to one side; to pour the melting products it is tilted to the other side.

A control panel on the working platform next to the converter was used to control the converter, to move the tuyere, to feed the blast and charge the materials from the hoppers. The movement of the hot-metal car was controlled from a panel on the shop floor level near to the rails of the car. When replacing the lining, the bottom of the converter was detached by means of a jack car with a lifting table.

The productivity of the continuously operating converter is assumed to be 1 million tons of steel per year. Allowing for the time spent on periodic overhauls, replacing the lining, etc., each converter of the three intalled in the shop has a productivity of 750-850 thousand tons of steel per year. The hourly productivity of the con-

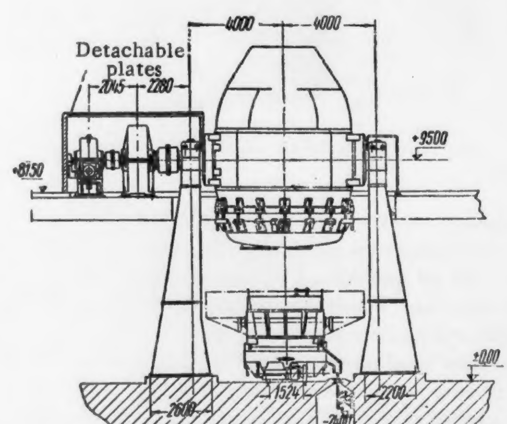


Fig. 1. 100 ton oxygen converter (test).

verter with the tested smelting time of 45 min is 120 tons. The oxygen consumption per ton of charge is 60 m^3 at a pressure of 10-12 atm.

Experience gained in operating the oxygen conversion process shows that the size and shape of the working volume has a decisive influence on the operation of the converter, the life of its lining, the smelting time, the splashing, the gas saturation of the finished steel, etc.

The converter has a cylindrical shape (Fig. 2). To improve circulation in the molten metal, the inner surface of the bottom is in the form of a concave sphere. The working volume of the converter is taken as 75 m^3. Operating experience showed that the specific volume, i.e.,the ratio of the converter volume to the weight of the charge, should be close to 1.0. When the converter operates with 100 tons of charge at the start of the campaign, the specific volume is 0.75, and as the lining is gradually burnt it becomes approximately 1.0-1.1, since the working volume of the converter increases to 100-110 m^3.

The depth of the converter bath is taken as 1.4 m when the lining is new, which means that toward the end of the campaign, when the lining of the lower part of the body is burnt to a certain extent, the depth becomes about 1.1-1.2 m. This depth is confirmed by operating experience at the Krivoi Rog plant, where smelting with a small Bessemer-type bath led to intensive burning of the bottom, and smelting with a double charge at a bath depth of 1.1 m did not reduce the life of the lining and did not spoil the process.

The height of the working volume of the converter and the diameter of its throat across the brickwork were determined by their effect on the quantity and intensity of splashing of the liquid bath during blasting. The height from the bottom to the edge of the throat was taken as 8.65 m, which gives a height to diameter ratio of 2.15 when the working space is 4.0 m.

A throat diameter of 1650 mm provides convenient pouring of the iron. A large throat diameter would increase splashing of the slag and molten metal during the period of vigorous boiling of the bath and furthermore as shown by the investigation, would lead to a high degree of saturation of the bath with nitrogen due to air entering the retort.

The design specifies a lining consisting of three layers - a working, an intermediate, and a fixed layer. This means that the converter can be operated until the working layer is completely worn out without risk of burning the metal shell. The working layer of the lining, which comes into direct contact with the molten metal, is made of periclase-spinel brick.

Of the many materials tested at the Petrovsk and Krivoi Rog plants, "PSh" brick was found to be the most suitable for the oxygen process. The thickness of the working layer varied: in the zone of greatest burning of the lining it was 690 mm and in the upper part of the converter it was reduced to 460 mm. The intermediate layer of tar-magnesite is a mixture of 90% of magnesite powder (2-6 mm fraction) and 10% of coal tar. This mass is packed-in hot during the installation of the working layer. The fixed layer is 115 mm thick and consists of highly refractory magnesite brick. The space between the fixed layer and the shell is filled with a material of the same composition as the intermediate layer. The total thickness of the lining at the lower and middle parts of the shell is 900 mm, the lining of the top section is about 600 mm.

The detachable bottom of the converter has a concave spherical working layer which is 460 mm thick and

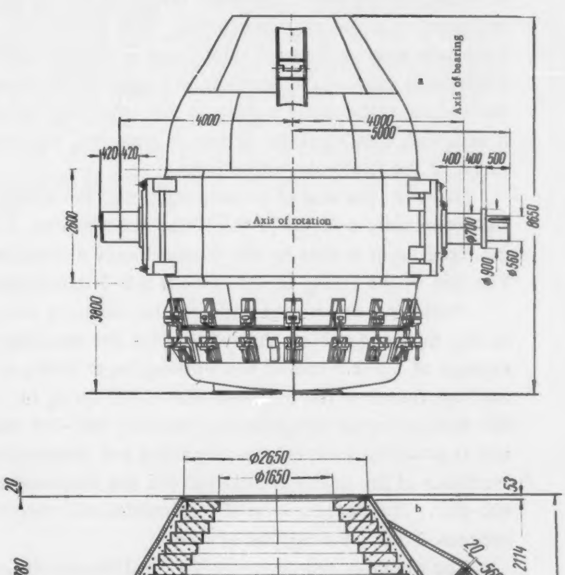


Fig. 2. 100 ton converter. a) Front elevation; b) section.

several layers of magnesite and fire (bottom) bricks. The total thickness of the bottom* is about 1000 mm.

The converter lining requires 250 tons of brick (206 tons of periclase-spinel, 34 tons of magnesite and 10 tons of fireclay) and 45 tons of tar-dolomite mass.

For the lining, 15 types of brick were used. Of these, 11 types were periclase-spinel, 3 were magnesite and 1 was fireclay. The central location of the converter throat considerably simplifies the lining of the top section and reduces the number of types of brick which must be used.

The body is lined "dry" with the bottom removed, using a special telescopic table which rises as the rings of lining are applied. The lower stages of the lining are collected on a flat portable jacket. The bottom ring of the bricks rests on a metal ring which is welded inside the bottom edge of the housing. All rings of the lining are wedged sufficiently tightly so that when the jacket is removed, and until the bottom is installed, the lower stages of the lining do not collapse.

Toward the end of a campaign, only the working and intermediate layers of the lining are renewed, since the fixed layer is able to last through many campaigns. The life of the lining on the bottom is 2-3 campaigns.

Operating experience with smaller capacity converters having top oxygen blast showed that for one smelting an average of 1.5-2.5 mm of the working layer lining is used up; therefore the proposed life of the lining for a 100-ton converter comprises not less than 300-350 heats, and as production experience is gained and intermediate overhauls of the lining are carried out this increases to 400-450. The consumption of refractories will then vary between 6.5 and 8.5 kg/ton of steel.

The planning requirements for the 100 ton converter and its auxiliary equipment prevented the use of the design solutions of existing converters in the design work. It was necessary to use an entirely new approach to the solution of the many design problems relating to the converter.

The dimensions of the shell at the bottom of the converter were determined by the inner working space and the shape of the 3-layer lining. The full height of the converter was 8650 mm, the maximum internal diameter across the shell was 5960 mm, the distance between the axes of the support bearings of the converter trunnions was 8000 mm.

The steel shell of the main section and the bottom is of welded construction of forged and bent sheet, which considerably simplifies the manufacture. The suitability of welded construction has been confirmed by operating experience with welded converters of small capacity and large steel-casting ladles.

The design includes the mounting of the converter on support bearings without a heavy cast support ring. This considerably simplified the construction of the converter since it dispenses with the casting of a complex steel ring weighing not less than 100 tons and having a diameter greater than 6 m, it reduces the weight of the converter, reduces the cost of manufacture and improves the cooling conditions during operation. Furthermore, it makes the converter more compact.

The trunnions are forged in one piece with the plates, and are fastened by a bolt connection to the beams which are welded to the central plate of the main section. Between the trunnions and the main section there is a clearance sufficient for efficient cooling of the shell. According to the proposed arrangement, the load from the weight of the converter and its tilting moment are directly transferred to the central belt of the main section, clearing the ring.

The shell of the main section, carrying the bearing trunnions, is made of 100 mm sheet steel. The top section and the bottom bell of the main section and also the shell of the bottom are made of 50 mm sheet steel. At the bottom of the top section, the main section of the converter is fitted with a tapping chute for pouring the finished steel. The outer surface of the main section is made without protruding ribs and plates, and it is made as smooth as possible to prevent the formation of incrustations.

The converter has a rotation mechanism with an electric drive from two simultaneously operating 95 kw/dc electric motors. In the event of one of the electric motors failing due to overload, it is possible to complete any given tilting operation and return the converter to the vertical position. The nominal speed of rotation of the converter is 1 r p m. The operation of the motors according to the system "generator-motor" makes it possible when necessary to control the tilting speed in descent within the limits 1:10.

It was thought previously that a converter with the motors switched off and the brakes failing should return to the vertical position from any angle under the action of its own weight and that the tilting of the converter should not be accompanied by a change in sign of the torque on its drive trunnion. The complete reliability of modern electrical brakes and the proposed mechanism make it possible to achieve the maximum reduction in the power of the drive and to avoid these situations.

The converter rotation mechanism was first planned in Soviet practice with closed reduction gears and with the use of a globoid worm gear for greater compactness. This arrangement prevents the vibrations of the converter and its bed from affecting the operation of the gear transmissions, since there is no end-mounted gear on the converter trunnion and the rotation mechanism is connected with the trunnion by means of a clutch, compensating possible tilting and displacements of the shaft axes. The mechanism consists of two reduction gears: a single stage cylindrical unit with herringbone gears and a two stage worm-cylindrical unit; the mechanism is situated on the working platform of the

*Editor's note. Since the wear of the bottom in converters with top blasting is very small, it is not necessary to make the bottom lining 1 m thick. Reducing this thickness leads to a reduction in weight of the bottom and makes it possible to reduce the power of the jack car.

converter section, it is protected from spray, and metal and slag sparks, and it is accessible for servicing during the operation of the unit.

The oxygen is fed into the converter through a water-cooled tuyere, moving over the converter along its vertical axis and consisting of three concentrically fixed steel pipes and a removable nozzle of fire-refined copper. The oxygen passes along the central pipe and through the jet of the nozzle into the reaction zone of the retort. Flexible pipes are used to apply the oxygen and cooling water to the tuyere.

The tuyere is fixed on a carriage which moves along guides by means of an endless chain. The movement is provided by a 12 kw/dc electric motor. The tuyere travel is 12 m; the extreme upper and lower positions are fixed by means of control apparatus, and its position in the converter along the height is recorded by a selsyn-transmitter and selsyn-receiver mounted on the control post.

The water consumption for cooling the tuyere is about 100 m^3/hr at a pressure of 8-10 atm. A special telpher crane moving along the line of converters is used to replace the burnt out tuyeres.

The steel is transported in a powered metal car with four axles, the drive axles of the car being of welded design, and a welded 90 ton ladle. With a yield of 0.87-0.89 of the metal charge, the ladle is able to receive all the finished material from the converter. The electrical power for the motors is provided from trolleys situated in a tunnel. For the transportation of a standard slag container of 16 m^3 capacity, there is a double axle trolley of welded design with self-coupling mechanisms.

To replace the converter bottoms there is a jack car with a telescopic hydraulic plunger, raising the car table; its capacity is 165 tons. The oil jack provides lifting and clamping of the bottom to the converter. The trolley has four axles and is not powered; it has a welded frame and lifting table and a cast hydraulic jack. The hydraulic system with pumps and electric motor is mounted in the frame of the car and is protected from the external atmosphere. A flexible cable with a plug joint is used to feed the electric current to the motor.

INCREASING THE OUTPUT OF CASTING BAYS

D. I. Deineko and N. V. Zaveryukha

Magnitogorsk Steel Combine

In the majority of large steel plants, the casting bays of the open-hearth shops represent one of the bottlenecks limiting the productivity of the open-hearth furnaces. It is particularly important to increase the output at the casting bays in shops with a large number of furnaces and with conversion of most of the furnaces to double charge.

This is particularly true of the No. 2 and 3 open-hearth shops of the Magnitogorsk Combine. At the present time in the open-hearth shops, 84% of the furnaces have double charge and tap the metal into two ladles; furthermore, many measures have been put into operation directed toward increasing the productivity of the furnaces: conversion to chrome magnesite roofs, increasing the number of casting machines, charging and casting cranes, increasing the volume of the molds, improving the preparation of the charge, organizing production and overhauls, etc.

As a result of these measures, the total melting of steel was doubled in comparison with that for 1945 and there was a considerable reduction in the time of the melt and time lost in overhauls, as can be seen from the following data:

Index	1945	1957	Reduction, %
Heat time, hours:			
at furnaces with single charge	14.42	9.15	36.5
at furnaces with double charge	16.7	13.25	20
Time lost in overhauls, %	17.1	7.1	by 2.4 fold

With increase in the productivity of the shops there was a continuous increase in the production load on the casting bays. After converting 40% of the furnaces to double charge, the equipment of the casting bays was still insufficient for casting the metal and supplying the furnaces with ladles and containers.

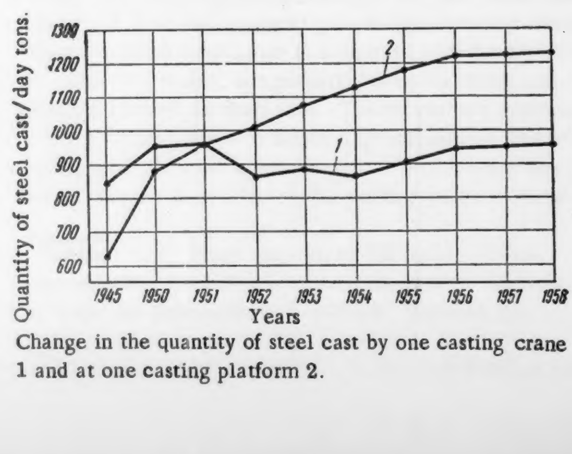
Increasing the number of casting cranes, steel-casting ladles, slag cars and slag containers reduced the load on one crane (see diagram). During this time there was no increase in the number of casting platforms; the load on each platform therefore increased continually and in 1956 it was twice the figure for 1945.

The large load on the casting platforms even with a sufficient number of casting cranes caused a large number of delays in the discharge of heats, frequently expressed in a retardation of the final melting. Further increase in the number of casting cranes cannot therefore give the required effect. It would be necessary to either increase the number of casting platforms or to accelerate the casting of the heats without spoiling the metal quality.

Under the conditions which obtain at the Combine, the casting bay length cannot be extended by more than 10%, which is quite insufficient. Another method was therefore employed at the Combine, i.e, acceleration of the casting. For this purpose, the heats of carbon steel from furnaces with double charge were poured simultaneously from two ladles into one pattern. In this casting method, the first half of the pattern is cast from one ladle (starting with the first mold) and the second half from the second ladle (starting with the middle of the pattern). The casting time for one pattern was halved in comparison with that for the conventional casting.

An essential condition in this casting method is the agreement of the chemical composition of the metal in each ladle with the limits fixed by GOST.

The difference between the carbon contents in the two ladles should not exceed 0.02% and the manganese and silicon differences should not exceed 0.03%. With this difference in the content of the elements, the variations in the mechanical properties of the metal do not exceed the permissible limits. The specifications tag of a melt indicates from which ladle a particular ingot was cast. However, this change in the casting technique did not fully remove the disproportion between the productivity of the furnaces and the productivity of the casting bays.



Change in the quantity of steel cast by one casting crane 1 and at one casting platform 2.

The Output of Casting Platforms with Various Types of Casting

Casting one composition	Casting time for one composition, %	Extent of use of one casting platform, %
From one ladle with one stopper.	100	100
From one 2-stopper ladle. .	70	87
From two 2-stopper ladles. .	40	73.5

For further reduction in the casting time, the steel casting ladles were fitted with second stoppers for simultaneous casting through two vessels. The springs and axles of the casting cars were strengthened in order that each could take four molds.

Transition to 2-stopper casting required a doubled consumption of steel-casting vessels, couplings and stoppers. Extra dryers were installed and old ones redesigned. The introduction of 2-stopper casting required the redesign of the steel pouring chutes in order to change the direction of the metal stream entering the ladle.

The large scale 2-stopper casting of rimmed steel began in August, 1953, and casting of killed steel began in August, 1954. Toward the end of the year the transition to the new casting method was complete and since January, 1955 all metal has been cast from 2-stopper ladles only.

The adoption of new casting methods, using two stoppers, and from two ladles in one pattern has considerably improved the speed of the process at the casting bays. The data given in the table show that the speed of working at the casting platforms has increased by 26.5%, and the casting time for one pattern has been reduced by 60%, i.e., by more than half.

This increase in the production of the casting platforms has made it possible to reduce the speed required in casting of ingots and to pour high-quality rimmed and killed steels through a nozzle of diameter of 30 mm instead of 35 mm and in this way to reduce the rejects due to cracks and flaws.

As well as increasing the production of the casting bays, the use of the new casting methods has provided a number of other advantages. The temperature of the ingots entering the blooming pits has been increased by 40°. Thus, heats of rimmed steel supplied to the pits within 1 hr and 10 min after the end of casting have the following average temperature for the same time of stripping depending on the casting method:

a) when poured from one ladle with one stopper, 890°,

b) when poured from one ladle with two stoppers, 910°,

c) when poured from two ladles with two stoppers, 930°

The life of the ladles has been increased by 20-30%. It has been found possible to control the speed with which the molds are filled, improving the ingot quality without increasing the total time for casting the metal from the ladles.

The productivity of the open-hearth furnaces is affected by their supply of slag containers. At the Combine this problem is mainly solved by increasing the number of slag cars and containers, improving the transport organization and tilting the containers, i.e., accelerating the cycle for the slag trips. In each open-hearth shop there are two slag-carrying trips for 8 slag cars. Initially the slag containers were mainly tipped at the slag dumps and the slag-carrying cycle took an average of 4 hours. The shop received 32 containers per shift. At the present time the slag is poured in the slag yard of the forging shop, thereby reducing the cycle to 2 hours and the shop now received 64 containers per shift.

The output at the casting bays is still nevertheless unsatisfactory. The open-hearth furnaces, supplied with highly refractive materials and fired with high calorie fuel, have still greater reserves of increased productivity. It is therefore essential to find ways for further increasing the operational speed of the casting bays. The most promising method for considerably increasing the output at the casting bays in existing shops, in our opinion, is pouring at high speeds through 60-70 mm diameter nozzles. A number of Soviet and other plants use this method and the quality of steel obtained is perfectly satisfactory. With this method, the speed of the steel casting has been doubled.

The use of this method requires the development and introduction of a new technique for melting and casting the steel, the main features of which are:

a) Low sulfur content in the steel;

b) lower temperature of the tapped metal (by 40-50°) than in casting through a 30 mm vessel; the permissible temperature limits are then narrower than when pouring through a 30 mm nozzle;

c) for rimmed steel, accurate control of the oxi-

dation of the metal during tapping (sometimes by the addition of aluminum).

Experiments were conducted at the Combine on the pouring of steel through a 70 mm nozzle. At present these experiments are being repeated Preliminary results indicate that with this method it will be possible to provide the same quality of surface on the ingots as is the case with a 30-35 mm nozzle; however the technique requires careful working out and the whole of the shop personnel must be trained in the technique.

INCREASING THE DURABILITY OF MOLDS

Yu. R. Mesnyaev and T. S. Konovalova

Verkh-Isetskii Steel Plant

The Verkh-Isetskii Steel Plant melts electrotechnical silicon steel in basic open-hearth and electrical arc furnaces. The steel is bottom poured, the ingots weighing 530 kg, the molds being open and wider at the bottom, without shrinkage heads; they are coated with marshalit solution* and are mounted in channels on 16-and 8-section trays. In winter the molds are cooled in air and in summer they are cooled with water.

The main reasons for the failure of the molds are longitudinal cracks at places near the joints of the mold boxes during casting; another cause is erosion during the pouring of open-hearth steel.

The molds are cast in the foundry from a cupola with 1300 mm throat diameter into semipermanent molds with the seams sealed with fireclay and then dried with a blow torch. The bottoms of the molds are reinforced with steel rings to increase the durability.

The composition of the charge is 40% cast iron, 30% conversion pig iron and 30% mold scrap. When using low-silicon cast irons, the charge is supplemented with pig iron or 45% ferrosilicon.

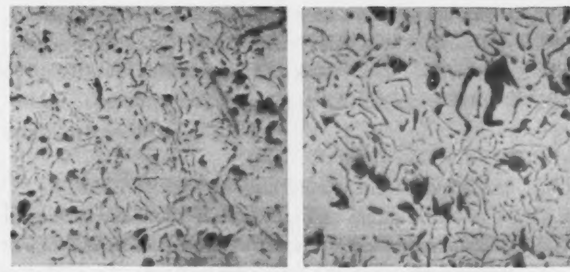
The chemical composition of the iron of the molds, %:

C	Si	Mn	P	S	Cr
3.4-4.0	1.5-2.5	≤1.25	≤0.25	≤0.10	up to 0.30

The molds have a perlite-ferrite structure with rather coarse unevenly elongated and randomly orientated graphite inclusions.

The casting technique had serious faults which reduced the durability of the molds. The appearance and development of longitudinal cracks along the lines of the mold box joints occurred due to the insufficient drying of the material used to seal the mold box seams and the insufficiently tight fastening of the mold box clamps. The moisture content in the seam after drying varied between 7 and 12%. Structurally free cementite was detected in the microstructure of specimens of the outer surface of the mold in the case of insufficient drying of the seam.

Poor fastening of the mold boxes with clamps led to opening of the seams (up to 8-12 mm) during the first hour after pouring due to expansion of the iron. At these points the cast mold was subjected to intense cooling, which is confirmed by the reduction in size of the graphite (Fig. 1). The graphite in a specimen from the surface of the mold on the outside of the mold box joint



a b

Fig. 1. The microstructure of an iron specimen of molds from the outside joint of the mold boxes (a) and the inner joint of the mold boxes (b).

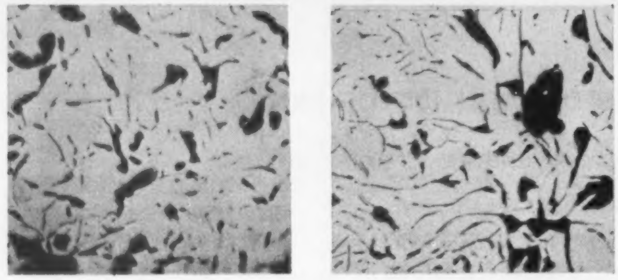
was finer than in a specimen from the side of the inside joint of the mold boxes.

Reinforcing the bottom of the molds with welded rings introduced a complication, in that during casting the ring took up an arbitrary position and often approached the lugs. Placing the ring close to the inside surface led to the rapid appearance of fine cracks near to the ring and to chipping of the iron. This led to jamming of the ingots and destruction of the molds. To increase the durability of molds in the casting shop, tests were carried out on an iron modified with 75% ferrosilicon in the casting basin (using the method of the "Zaporozhstal'" Factory); after dismantling the mold boxes, slow cooling was carried out, top pouring was used, reinforcing rings were placed around the external perimeter, the seams were dried and the mold heated with a special charcoal burner using forced blast and using quick drying cement consisting of core material and sulfide liquor, etc.

To obtain iron with a more even temperature and chemical composition, together with increased carbon content (up to 3.80%) the cupola diameter was increased to 1500 mm. The test molds were cast in different varieties (11 in all) and during operation they were observed continually. The results of the observations indicated that the technique for casting molds should be changed.

Modifying the iron on the trough with 75% ferrosilicon in quantities of 0.5-0.9% proved ineffective and it was replaced by feeding FeSi into the ladle onto the purified specular pig iron. The molds modified according to the first method took an average of 58 casts, those

*A mold wash containing quartz powder.



a b

Fig. 2. The microstructure of unmodified (a) and modified (b) iron.

modified by the second method took 68 casts.

Two batches of molds cast without modification were cooled slowly after the mold boxes had been dismantled. After removal of the mold boxes they were cooled in a pit for several days. The slowly cooled molds could take an average of 26-34 casts.

The molds were top cast through a slot gate and could take an average of 42 casts. In this casting method there was a considerable amount of contamination at the inner surface due to erosion of the core and mold. The main reason for the failure of molds cast with this method were longitudinal cracks. The slow cooling and top casting method was therefore not put into production.

To check the effect of the charge on the mold durability, experimental batches were cast from charges of varying composition. Two batches were cast from charges containing cast irons without modification and without the addition of ferrosilicon. The molds of these batches lasted through 74 and 68 casts respectively.

To determine the effect of the new casting technique on the mold durability, one batch was cast with a normal charge consisting of 40% cast iron LK-4, 30% conversion cast iron and 30% of mold scrap. The molds of this batch lasted 77 casts whereas those cast from this charge but with the old technique only lasted 57 casts.

Thus, improving the casting technique by improving the drying of the molds, reinforcing the bottom of the molds with rings around the external perimeter and modifying the iron in the basin with 75% ferrosilicon, even without changing the charge composition, improved the mold durability by 35%.

In the case of molds cast with a modification in the casting basin it was found that cracks formed later (after 50-60 casts) and are distributed evenly across the section of the mold, but not along the joint of the molding boxes. A comparison of the microstructures shows that with modification the graphite inclusions are distributed evenly, have a rounded form and the filaments are reduced (Fig. 2).

A study of the molds of other casting batches shows that with increase in the content of the conversion iron in the charge from 50 to 100% their durability decreases somewhat. Up to 70% of the molds for casting open-hearth steel have cavities on the inner walls due to erosion from the metal stream during filling. The reasons for the erosion are the heating of the metal above 1650° and the casting speed being increased above 0.47 m/min.

To improve the operating conditions, the molds were given a double coating of marshalit solution, the casting speed for electrotechnical steels of all types was kept between 0.35 and 0.47 m/min and the soaking time of the ingots in the molds during the casting of open-hearth steels was reduced to 1 hour and in the casting of electrical furnace steel to 1 hour 20 min.

Due to improvement in the technique, the mold durability was increased with a simultaneous reduction in steel consumption from 31-29 to 26-25 kg/ton of smelted steel. From a charge having up to 50% low-silicon iron, it is possible to obtain molds with satisfactory durability (74-77 casts).

It was found that with molds made without cast iron from a charge consisting of 50% conversion pig iron and 50% mold scrap, the durability is reduced by 12%, and for molds cast from a charge with 60-100% conversion iron it is reduced by 25%.

FRUITFUL SCIENTIFIC AND TECHNICAL COOPERATION

S. G. Afanas'ev

In May, 1959, at Dnepropetrovsk the first Conference was held of the Working Group for Converter Production of the Permanent Commission on Ferrous Metallurgy formed by the member countries of the Council of Economic Cooperation. The Conference was attended by steel workers and refractory workers from Bulgaria, Hungary, East Germany, Poland, the USSR and Czechoslovakia. The delegates inspected the converter shops at the Dzerzhinskii, Petrovsk, and Krivoi Rog Plants.

Lectures and discussions were held on the operating experience of converter shops and research work in the field of converter production.

The next few years will see a considerable increase in the melting of converter steel in many Communist countries. Thus, in Bulgaria, which has the rich Kermnikovsk iron ore deposits, iron will be melted with 4-8% manganese content. Calculations and experiments conducted in the USSR have shown that this iron is best converted with top oxygen blast.

When blasting iron with an average Mn content of 5.7%, a slag is obtained with 60.5% MnO. Experimental melting was carried out by TsNIIChM*; the amount of this slag comprises 7.2% of the weight of iron. Manganese slag is used in the melting of ferroalloys. The semifinished product containing 1.9-2.3% C and 1.0-1.2% Mn (after tapping the slag and adding lime) is blasted until metal of the required composition is obtained.

The metal quality is better, the sulfur content is 0.009-0.020%, phosphorus 0.020-0.030%, nitrogen 0.004-0.005%. Good results were obtained in deep drawing tests.

In Bulgaria, work will soon start on a modern converter shop with oxygen blast for processing iron with high manganese content.

Oxygen-enriched blast is used in the basic shop of the Markshütte Plant (Unterwellanborn) where there are 4 converters with a 17.2 ton charge. The oxygen consumption is 18-22 m^3/ton. I. Olden, a research worker from the Freiberg Mining Academy, reported on work aimed at improving the quality of basic Bessemer steels. At the present time, the basic steel smelted in East Germany is used in construction work, rolling stock construction and in the production of rails. Considerable improvements have been made in the plasticity of the steel and in reducing "aging" by reducing the nitrogen in the metal and improving the method for deoxidizing the steel.

T. Shobel of the Freiburg Mining Academy presented an interesting report on the production of pitch-dolomite refractories for the converters of the Markshütte Plant. The 380 mm converter linings last more than 200 heats, and the bottoms last for 38.6 melts. Improvements in the granular composition of the dolomite made it possible to increase the bottom life to 44.5 heats.

A good dolomite should have the following chemical composition, %.

SiO_2	R_2O_3	CaO	MgO	Remainder
0.46	0.62	30.98	22.90	45.04

The roasting of the dolomite is an important factor in improving the quality of the dolomite lining. For improved impregnation with pitch, the roasted dolomite should have a porosity of about 20%. Dolomite roasted in rotary furnaces is difficult to impregnate since its pores become blocked during the roasting. The dolomite is therefore best roasted in shaft furnaces.

A new converter shop is being planned in East Germany and may be built at the plant in Stalinstadt. In this work the German engineers are being helped by the Planning and Research Organizations of the USSR.

The Polish delegate, S. Tochowicz, reported on results obtained in the melting of steel in 10 ton converters with side blast iron containing 1.56-2.60% Mn and 0.85-2.02% Si. The oxygen consumption was 700 m^3/ton of iron. Additions to the charge were 6-8% lime, 0.15-0.20% fluorspar and 1.0% clinker (of the weight of iron).

In experimental melts a high degree of dephosphoration was achieved, averaging 83.7%, the best phosphorus removal being obtained from iron where the silicon content did not exceed 1%. The maximum degree of desulfurization was also achieved with these melts (averaging 54.36%). The nitrogen content in the steel varied from 0.0017 to 0.0046% (an average of 0.0027%).

The most difficult problem is that of increasing the durability of the refractory converter linings, especially in the tuyere zone.

In Poland at the Lenin Plant in Nova Huta a large converter shop is being built with oxygen blast; this shop will have 100-ton converters. The Conference therefore dealt with the design and operating features of 100-ton converters.

The Soviet delegates, S. G. Afanas'ev, N. I. Beda, I. S. Kukuruznyak, and V. D. Umnov, reported on operating experience at the converter shops of the Petrovsk and Krivoi Rog Plants and on research work into con-

*Central Scientific Research Institute of Ferrous Metallurgy.

verter processes. More than two and a half million tons of converter steel is being smelted in the USSR, using oxygen blast and a considerable amount of experience has been gained in the study of converter processes, improving the metal quality, and planning converter shops. Soviet steelmakers are helping the metallurgists of the Communist countries to increase steel smelting in converters.

Basic converters are operated in Czechoslovakia at the Kladno Plant. There they have a 5-ton experimental converter in which studies are made of the application of oxygen blast.

The Czechoslovak metallurgists are interested not only in the possibility of converting phosphorus irons in the converters with oxygen blast, but also in smelting converter steel from chrome irons (using Albanian ores).

In Czechoslovakia they are planning to build converter shops which will operate with top oxygen blast.

Of considerable interest was the paper by the Czech engineer L. Kodrle on the study of temperature in oxygen converters, using 12 thermocouples in metalloceramic protective tubes placed in 5 different zones of the converter lining.

It was found that the change in temperatures showed the same character and did not depend on the initial temperature of the iron poured into the converter. The highest temperatures developed over the level of the bath and at the level of the bath surface. The temperature changes at various levels of the bath make it possible to follow the circulation of metal in the bath. In the first period of blasting, the metal moves along the walls to the top, and in the middle to the bottom and then in the opposite direction.

An interesting report was presented by Ya. Plieshovskii, a worker at the Refractory Institute in Bratislava, on the manufacture of refractories for lining converters operating with oxygen blast.

Recognizing that the converter method for smelting steel will be extensively developed in the Communist countries, the Conference passed a resolution calling for the systematic interchange of information and results on research work in converter production.

ROLLING LIGHT-SECTION BEAMS

M. M. Shernov
Senior roll designer at the Magnitogorsk Metallurgical Combine

At Mill 500 of the Magnitogorsk Metallurgical Combine the rolling of No. 12 and 14 light-section beams has been mastered.* The first beam is in regular production, the second one has been tested in an experimental run.

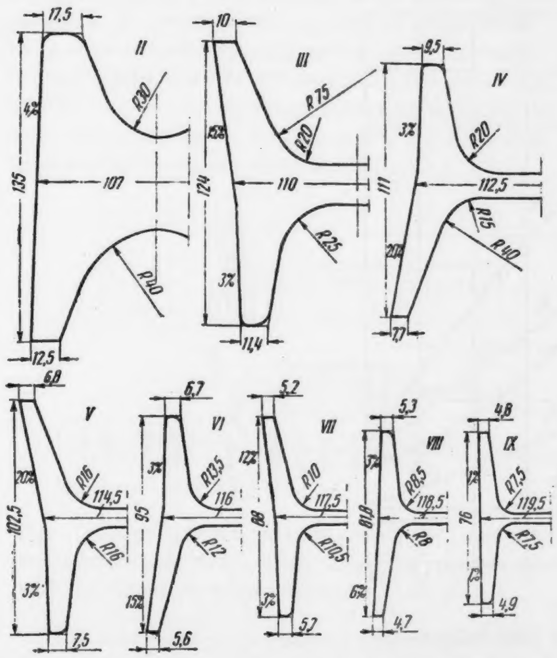


Fig. 1. Dimensions of the templets for rolling light-section beam No. 12; II)-IX) stands.

Because of the inadequate number of stands at mill 500 (each beam is shaped in eight passes), it is necessary to apply high drafts which greatly exceed the recommended values. To make the high-draft rolling process easier and to prevent the formation of laps at the base, the flanges are bent out at large angles which in some passes reach 20% (Fig. 1). At the same time, all the other advantages of employing bent-out flanges are made use of.

Unlike the bent-out type of passes used at other Plants, the H-beam passes at Magnitogorsk Metallurgical Combine (Fig. 1) have a straight web which makes the machining of the rolls easier and also simplifies the construction, installation and operation of rolling accessories.

In fact, as was found by the analysis of the operation of the pass (Fig. 2) and the study of the defective sections, the web of the section bends when it enters the pass since the rigidity of the base of the flanges is considerably higher than that of the web. The bending of the web and the subsequent straightening, as well as the preliminary bend of the web (Slick† Method) used at other Plants for rolling in bent-out type of H-beam passes does not present any additional difficulties in operation.

The dimensions of the templates were calculated on the basis of a standard chart of the ratios of the deformation coefficients for each element of the H-beam groove in various passes (Fig. 3). The actual deformation coefficients in some passes are slightly different from the ones indicated on the chart because of some more accurate determinations and rounding-off, but in general, in the pass arrangement for rolling H-beams on Mill 500 the character and the ratios of the dimensions are maintained fairly accurately within the specified limits.

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* Roll designer Eng. N. I. Sirazitdinov took part in the design of the rolls.

† Name not verified – Publisher's note.

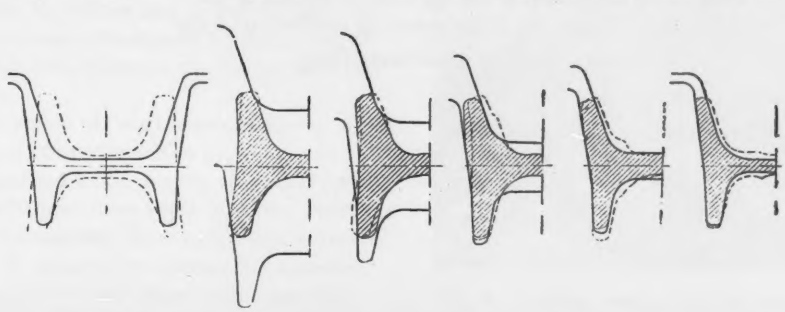


Fig. 2. The deformation process of the section in the H-beam passes with bent-out live flanges.

The chart shown in Fig. 3 was obtained as a result of the analysis of a large number of pass arrangements and practical observations, and it can be recommended as a standard for an H-beam pass design. Of course, for beams of other dimensions and for other mills it is necessary to take different absolute values of the deformation coefficients but their ratios and the general trend of the curves should always, we think, be as given in Fig. 3.

The deformation coefficient of the height of the section for dead flanges is more than one (1.07-1.15), and in live flanges it is equal or even slightly smaller than one. Therefore if the passes are designed in accordance with the chart (Fig. 3) the draft coefficients of live and dead flanges and the web are practically equal in the last 2-4 passes (depending on the absolute values of λ). In the initial roughing passes, the draft coefficient of the live flange is greater than the draft coefficient of the dead flange.

Further investigations are necessary to determine the optimum ratios of the deformation coefficients for various elements of the H-beam pass accurately.

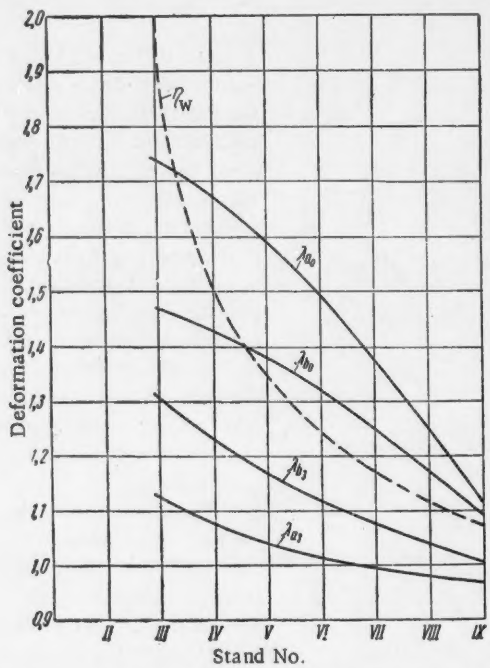


Fig. 3. Chart for selecting the ratio between the deformation coefficients in the grooves for various passes: η_{ω} is the deformation coefficient (reduction in thickness) of the web; λ_{a_0} is the reduction coefficient of the thickness of the end of the live flange; λ_{b_0} is the reduction coefficient of the thickness of the base of the live flange; λ_{a_3} is the reduction coefficient of the thickness of the end of the dead flange; λ_{b_3} is the reduction coefficient of the thickness of the base of the dead flange.

THE MECHANIZATION OF THE PIECE-BY-PIECE FEEDING OF STEEL SHEETS TO THE OILING MACHINE

A. P. Koshka and V. A. Brusilovskii

Novosibirsk Metallurgical Plant

For the prevention of corrosion of cold-rolled annealed steel plates, they are covered on both sides with a coat of linseed oil before they are packed.

The old oiling machine which was used in the cold-rolling shop at our Plant, consisted of the following main parts (Fig. 1): gravity rolling tables with metal rollers and a retractable stop; gravity rolling tables with soft composite rollers; stands with two oiling felt rollers one of which was the driving roller; frames for arranging the sheets into packs; an oil tank, oil piping and a manifold for supplying the linseed oil to the felt rollers.

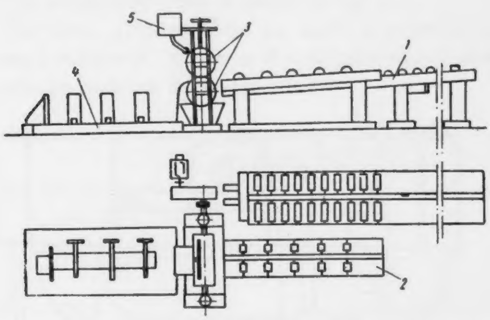


Fig. 1. Diagram of the old oiling machine: 1) gravity roller tables with metal rollers; 2) gravity roller tables with felt rollers; 3) rollers; 4) frame; 5) oil tank.

The oiling process consisted of several operations of which only one – the placing of the sheet packs onto the receiving roller tables – was mechanized. The remaining operations, i.e., the transfer of the sheets to the second roller tables and the entry of the sheets between the oiling rollers, the placing of the sheets in the frame and the supplying of the linseed oil to the tank, were carried out by hand.

The composite felt rollers used to wear rapidly, and hence, the machine had to be stopped frequently resulting in unproductive labor losses on the change and overhaul of the rollers.

The oiling machine has been modernized (Fig. 2) by the replacement of the felt rollers by metal rollers and by the mechanization of a number of manual operations. A semimechanical sheet feeder has been installed; it consists of pack-collecting roller tables with a damper mechanism (Fig. 3) actuated by a pneumatic cylinder via a lever system, and a pack turner with a movable frame (Fig. 4). The cradle of the turning mechanism supports a section of the roller tables (similar to the collect-

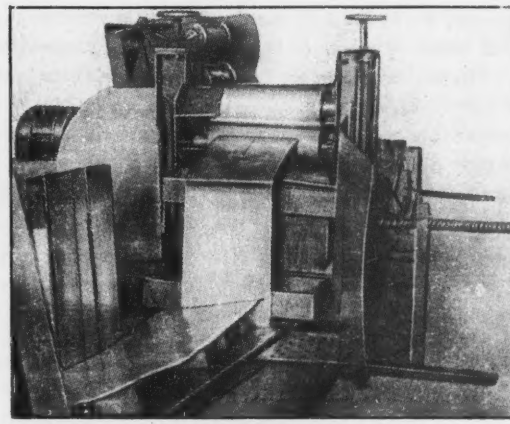


Fig. 2. Modernized oiling machine.

Fig. 3. Collecting roller table (Fig. 1), pneumatic cylinder (2) and levers (3).

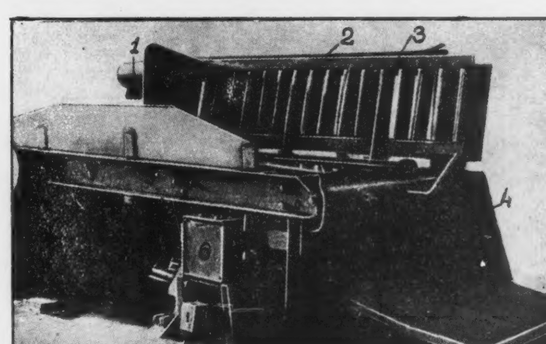


Fig. 4. Turning mechanism for sheet packs; 1) cradle; 2) collecting roller tables ; 3) toothed sector; 4) stand; 5) belt conveyor; 6) brakes.

ing roller tables), a removable frame and toothed sectors, and revolves on journals mounted in the bearings of the turning mechanism. The cradle with a sheet pack is turned by a 10 kw, 510 rpm electric motor equipped with four-speed gears. The drive is provided with electromagnetic brakes. The cradle can turn though an angle of 85 degrees; the accuracy of the setting is ensured by track switches; the angle at which the frame with the sheets falls down is 100 degrees. As the sheets are removed, the frame with the sheets is transported to the roller tables by means of two hand-operated toothed rails.

The belt conveyor for entering the sheets between the rolls is driven by the oiling roller by means of a chain transmission. When the sheet is entered into the machine, the pack is transferred from the collecting roller tables onto the cradle and then is turned through 85 degrees. Each sheet is then turned further by hand through 15-20 degrees and then it falls by gravity on to the conveyor which takes it to the machine.

Tests carried out at our Plant show that it is possible to mechanize the feeding of the sheets also by using rubber suckers for turning the sheets through 15-20°C. However, since during the annealing the sheets frequently become welded at the edges, the use of the rubber suckers necessitates a preliminary opening (tearing apart) of the sheet packs.

It should be pointed out that if it were possible to feed the sheets to the machine before the annealing, the application of the feeder with rubber suckers would make it possible to mechanize this extremely labor-consuming operation which is repeated several times during the finishing process for steel sheet.

The modernization of the lubricating machine made it possible to reduce manual labor considerably and to increase the operating efficiency in this section of the mill. The annual saving constituted 55,000 rubles.

MODERNIZATION OF THE 1500-TON HYDRAULIC PRESS

V. G. Attaryan

Mechanic of the tube billet mill at the Zakavkaz Metallurgical Factory

Tube billet mill 900 at the Zakavkaz Metallurgical Factory has a vertical hydraulic press of 1500 ton capacity designed by TsBKM (model PO-94) and designated for straightening and shearing in cold condition the tube billets of 160-360 mm diameter from steels of 100 kg/mm^2 tensile strength.

Before the billet is sheared a notch 4-6 mm wide, 20 mm deep through an arc of 60-90° is cut by means of a kerosene flame. If the notch is made by means of a hot saw the sheared surface of the billet is unsatisfactory and the shearing operation is difficult.

The modernized press is shown in Fig. 1.

The billet, arrested under the knife, is turned with the notch downward by means of manipulators mounted on carriages between rollers (Fig. 2.)

When the supports approach it, the billet is held by the lifting rollers mounted on the lower cross bar of the press between the outside rollers of the adjoining roller tables and the movable rollers; the rollers are lifted out and lowered by a pneumatic cylinder.

Before the modernization the press did not work satisfactorily. While the design capacity of the press was 80 shearing operations per hour, the maximum throughput which was achieved was 35-45 shearing operations per hour.

After the modernization of the auxiliary mechanisms carried out by our Factory, the throughput of the press reached 90-120 operations per hour on the average. The press is suitable for shearing tube billets only.

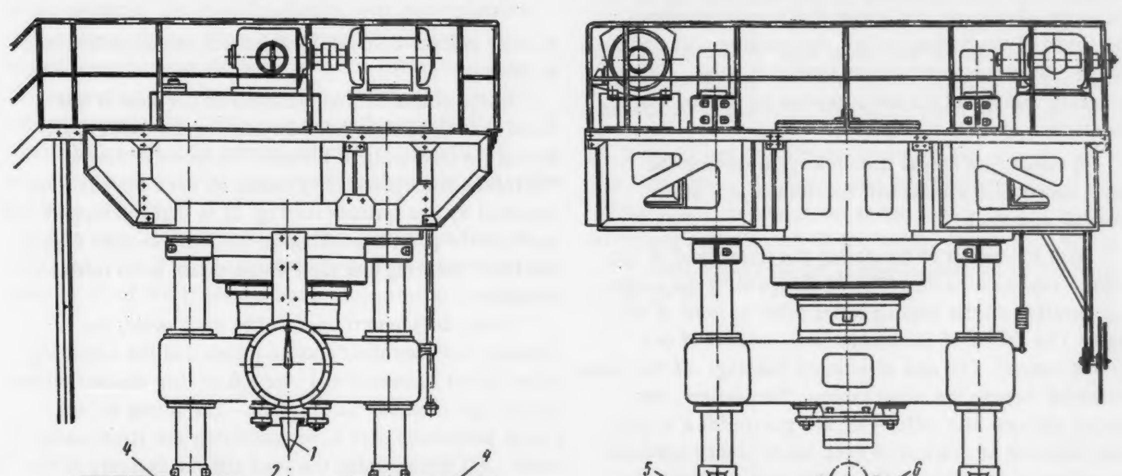


Fig. 1. 1500-ton hydraulic press after the modernization: 1) knife; 2) support; 3) spring; 4) guides; 5) lever; 6) shaped roller; 7) air cylinder.

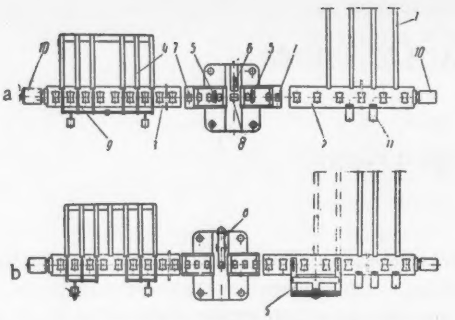


Fig. 2. The layout of the auxiliary mechanism of the press: a) before the modernization; b) after the modernization; 1) charging table with batcher; 2) feeding roller tables; 3) delivery roller tables; 4) pockets; 5) left and right manipulators; 6) middle manipulators; 7) lifting rollers; 8) central lifting mechanism; 9) pushers; 10) stop; 11) shock absorbers.

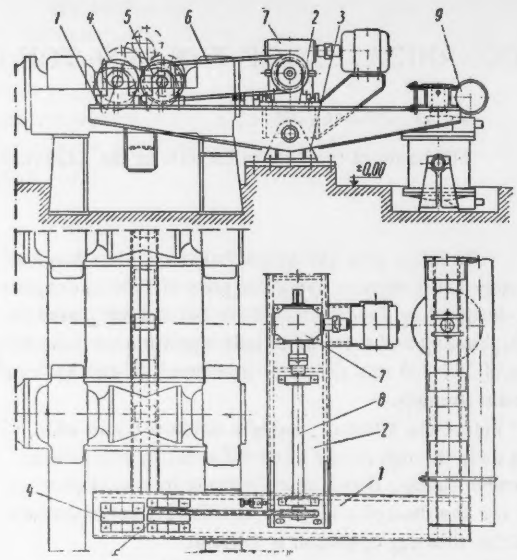


Fig. 3. Manipulator designed by V. G. Attaryan for turning tube billets: 1) frame; 2) carriage; 3) bearings; 4) nondriving roller; 5) driving roller; 6) belt; 7) gears; 8) transmission; 9) counterweight.

The main cause of the low throughput of the press was the unsatisfactory construction of the hydraulic manipulators. It was found in the course of operation that the manipulators could not turn billets of 230-270 mm diameter and broke down all too often; the naves of the worm wheels and the hosses of the gears fractured, the shafts of the turning rollers, the guiding rails and the lifting rods of the manipulators tended to bend. A considerable leakage of oil and scratches on the rod due to penetration of scale took place.

In effect, the actual time taken for turning the billet constituted almost half the total cycle of the shearing operation.

With the object of increasing the throughput of the press, a new manipulator (Fig. 3) designed by the author was installed on the feeding roller table in front of the press. The frame of the manipulator consists of two welded beams. The axis of rotation bearings of the frame is located outside the roller tables. Two rollers, one driving and one idle roller with the groove for a V-belt, were installed at one end of each beam placed between the rollers of the roller tables. The rollers have roller bearings.

The frame supports a carriage with a drive consisting of a 1 kw electric motor, worm gears, a transmission, and V-belts protected by welded guards. One end of the transmission shaft has a sheave for the belt. At the other end of the frame (between the beams), there is a pneumatic cylinder for lifting and lowering the billet. Turning a billet takes 8-12 secs.

The construction of the manipulator is simple and it is reliable in operation. All its mechanisms are located away from the roller tables and are not affected by scale.

When the billet is sheared the mobile supports move apart and since the hydraulic scheme had no provision for their moving apart during the billet shearing, the cylinder rods which were actuating the supports tended to bend and the bolts which supported the cylinders used to fracture.

In the course of the operation of the press it was found that there is no need for moving the stops apart during the fracturing of billets of various diameters. Therefore, the hydraulic cylinders on each support were replaced by two springs (see Fig. 1) so that the support could move apart depending on the force exerted during the billet shearing and then would return to its initial position.

When the supports approached each other, the distance between their outside rollers and the adjoining roller tables increased and the 1.2 m long sheared billets did not get onto the roller tables. The lifting rollers (lifted pneumatically) were nondriving and structurally weak. All this retarded the feed and the delivery of the billets. Therefore the lifting rollers were replaced by stationary rollers with a separate electric drive and for this purpose the frames of the adjoining roller tables were extended correspondingly.

Owing to the ingress of scale the plungers of the hydraulic shock absorbers frequently scratched and seized so that the press had to be stopped.

The hydraulic shock absorbers were replaced by springs. As a result, however, the rollers broke down more frequently. At the moment when the billets were sheared, the ends of the billets lifted with a jerk; the rollers actuated by the springs and the weight of the electric motor returned rapidly to their original position and hit against the side plates and frequently broke at the neck. For the prevention of sudden shocks, the supports were reduced to 5 mm (instead of 20 mm), the diameter

of the roll neck was increased from 80 to 110 mm and the roller bearings were replaced by cast iron bushing because the construction of the carriages did not permit an increase in the dimensions of the rollers suitable for new roller bearings. At the same time, the roller body was extended so that it was possible to use an electric motor with a stronger shaft.

As a result of all this, the stoppages of the press, which were previously due to breakages and defects of the movable rollers, were completely eliminated.

The shock absorbers, intended for protecting the movable cross bar from shocks and from billets which might fall from the movable rollers, broke down frequently because of the fracture of shafts or beams. The shock absorbers have now been removed and in their place guides for the prevention of billets from falling have been installed on the movable rollers (Fig. 1). The height of the knife was increased by 50 mm in order to prevent the billets from hitting against the upper cross bar. During the shearing of the billets, the block with the knife was gradually displaced in the direction of the billet entry and this caused frequent fractures of the fork joining the blocks to the movable cross bar. The displacement of the block was limited by plates fixed with short rods.

The central hydraulic lifting mechanism, mounted in the center of the press foundation, was unsatisfactory in operation because the scale entered the cylinder and caused scratches and the seizure of the lifting cylinder. The supporting groove of the lifting mechanism impeded the movement of the billets on the roller tables; the broken billet could be moved only by the next billet. At present a new lifting mechanism designed by the author has been installed.

The new lifting mechanism is made in the form of a welded lever (Fig. 1) one end of which is located on the center line of the press, and has a shaped roller mounted on roller bearings; at the other end of the lever is a pneumatic cylinder which effects the raising or lowering of the mechanisms.

The new lifting mechanism is not affected by the scale, and the roller (which replaces the groove) makes it possible to deliver a short billet easily without moving the next one, as was the case previously. The installation of this lifting mechanism made it possible to increase the throughput of the press and to strengthen its foundation by making it monolithic.

The length of the knife was increased by 200 mm so that it is now possible to use it twice; after one end is worn out, the knife is moved 200 mm along its axis and the billets are then sheared with the other half of the knife.

As a result of the modernization, the throughput of the press has increased from 35-45 to 90-120 shearings per hour; the stoppages, which were due to defects, breakages of mechanisms and leakage of oil, have been almost completely eliminated.

At present, the press not only provides all the required material for mill 400 but also shears the tube billets of 90-120 mm diameter from 36G2S steel* as well as processing rejected metal into scrap.

In the first half year of operation after the modernization, 25.4% more steel was sheared on the press than in the same length of time before the modernization. This experience should be taken into account in the design and manufacture of similar presses for other metallurgical works.

* Tube billets of diameter less than 160 mm are cut by the 1000-ton cold-cutting shears. If billets are cut from 36G2S steel, the end surfaces are uneven and have cracks at an angle to the axis of the billet. The billets of this steel shear easily on the 1500-ton press with a knife stroke of 10-20 mm.

NEW EQUIPMENT FOR THE COLD ROLLING OF SHEETS AND STRIPS

Eng. A. M. Kogos and Cand. Tech. Sci. É. R. Shor

TsNIITMASh

MILLS FOR ROLLING SHEETS OF VARYING CROSS SECTION

The use of equivalent strength sections of variable cross section (beams, tapering sheets, strips, etc., Fig. 1) makes it possible to reduce the consumption of steel without reducing the strength of the final products. Until recently, such sections were made by upsetting on presses and machining on special milling machines or by welding and riveting several strips of uniform cross section but of different thickness, and in some cases by etching metal strips in concentrated solutions of alkalis or acids.

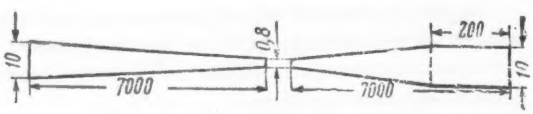


Fig. 1. Plates of varying cross section.

All these methods, however, have several disadvantages. During the machining or etching, the mechanical properties of the metal are impaired because the steel surface layer, which has a higher strength then the core, is removed. In addition, the fibers are cut during the machining and hence the strength of the metal is lowered; also the waste constitutes 40-50% of the weight of the initial material and consequently the cost of the final product is substantially increased.

TsNIITMASh, under the supervision of A. I. Tselikov, developed advanced methods of making section of varying thickness by means of special rolling techniques. A particular feature of mills for the productions of such sections is a continuous change in the distance between the axis of the rolls in the course of the rolling process (Fig. 2). By arranging a variable ratio of the speed of delivery of the sheet or the strip from the mill to the speed of drawing together (or moving apart) of the rolls, it is possible to obtain rolled pieces of varying cross sections.

For the production of strips and sheets, two-high or four-high mills with a powerful drive of the mechanism for the vertical displacement of the upper working roll are used. When the piece is entered between the rolls and is gripped by the rolls, an automatic signal is given for starting the electric motors which drive the screw-down mechanism and in this way the initial sheet or strip of uniform cross section is reduced in one or several passes to the product of varying cross section.

At present it is possible to obtain a taper of the

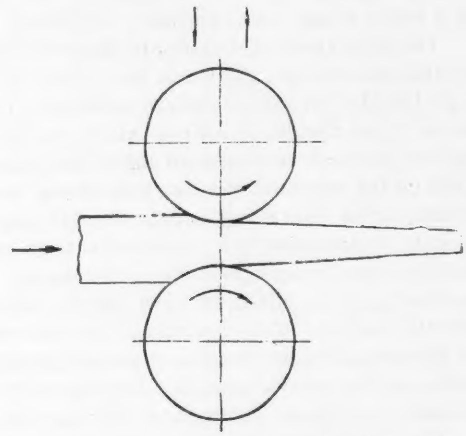


Fig. 2. Diagram showing the method of rolling the sheets of varying cross section.

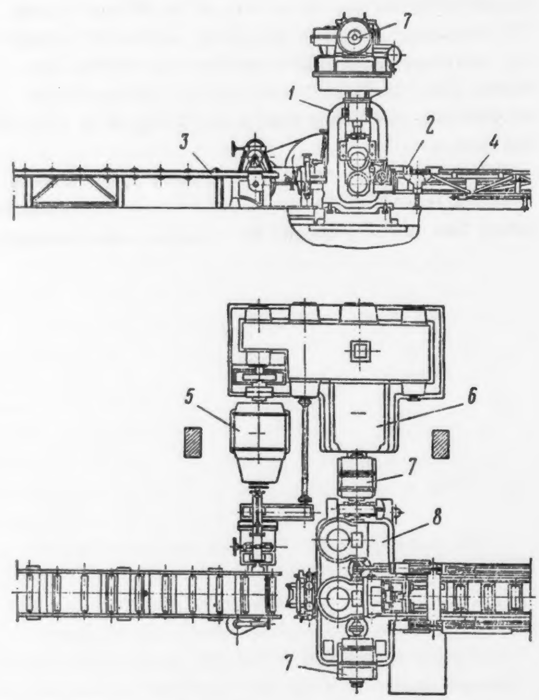


Fig. 3. Two-high mill 500 for rolling sheets of varying cross section: 1) working stand; 2) stretching mechanism; 3) and 4) entry and delivery rolling tables; 5) electric motor; 6) reducing gears; 7) electric motors of the screw-down mechanism; 8) screw-down mechanism.

rolled sheets and strips of varying cross section of up to 1.5 mm/m (considering the capabilities of the operating mills intended for this purpose).

The specific feature of the working stand of these mills (Fig. 3) is a very much higher power of the electric motors of the screw-down mechanisms as compared with the working stand for rolling plates of uniform cross section. The stretching mechanism, which serves for improving the delivery of the sheets and lowering the stress of rolling, is located at the delivery side of the working stand and constitutes a carriage which is moved on the guides by means of chains. Before the rolling operation, the carriage is brought near to the working rolls, and the sheet, after it leaves the rolls, is automatically gripped by the carriage which then begins to move and establishes the required pulling force.

For levelling sheets of varying cross section, two hydraulic cylinders which have plungers with collar bearings are placed under the screw-down mechanism. During the levelling process the screw-down mechanism does not operate and the required force on the rolls is obtained due to the pressure of the liquid in the cylinders. Since the plunger of the cylinder presses against the screw-down screw, the liquid entering the cylinder causes the cylinders to descend together with the upper roll in accordance with the taper of the sheet. At the same time, a constant rolling pressure is maintained; the pressure can be varied during the levelling by means of varying the pressure of the liquid in the cylinders.

The rolling of tapering sheets (initial materials are annealed sheets of uniform cross section) is carried out in a few passes with or without intermediate annealing depending on the magnitude of the total reduction at the thin end of the sheet. For convenience and efficiency, a whole batch of sheets are rolled with one and the same roll adjustment and then the sheets are transferred to the entry side of the mill and the roll is adjusted for the next pass. The sheets are then passed into the entry apparatus, and the screw-down mechanism is set up for raising the roll. The upper roll is set up in such a way that after the electric motor of the screw-down mechanism is revved up, the distance between the rolls corresponds to the required reduction at the thin end of the sheet.

At the moment when the required distance between the rolls is set up, the entry mechanism is started automatically; the support which holds the sheet is lowered and the sheet enters the rolls, where it is gripped by the rotating rolls and is passed through. During the rolling, the screw-down screws continue to rotate at a preset speed so that the distance between the roll increases continuously so that the final rolled sheet has a tapering cross section. After it leaves the rolls, the front end of the sheet is gripped by the carriage of the pulling mechanism. The tensile stress which assists the rolling is of the order of 0.2-0.3 of the yield stress of the metal being rolled.

After they leave the rolls, the sheets are collected on the delivery tables and are stacked into a pile while the screw-down mechanism and the entry mechanism are returned to the original position.

At the cold-rolling mill for wide sheets of varying cross section, the billets are delivered to the rolls by roller tables, and the drive of the screw-down mechanism is switched for raising or lowering the upper roll by means of a lever switch or a photorelay.

The thickness of the thin end of the sheet may be somewhat uneven but any irregularities are eliminated during the subsequent straightening of the sheets on a special stretching machine. The straightening of sheets with pronounced taper (for instance, with the thin end 1 mm thick and the thick end 4 mm thick) should be carried out on a roller-straightening machine which is of the same construction as ordinary sheet-straightening machines; the speed of the straightening is 20 m/min. The machine consists of an upper (movable) and a lower (stationary) welded housing each of which contains a set of straightening rollers: the upper housing has nine rollers and the lower one has eight. The diameter of the rollers is 90 mm and the distance between the rollers is 100 mm. Each row has four groups of free running back-up rolls.

The speed of the screw-down mechanism at the entry and delivery end of the straightening machine is regulated within the range of 1:10 by the adjustment of the terminal voltage of the electric motors supplied from separate generators. The straightening rollers revolve continuously during the straightening of the sheets of varying cross section. The sheets enter the machine thin end first (the machine can also be set up for receiving the thick end of the sheet first). The machine operates automatically.

The mills for rolling sheets of varying cross section are sometimes equipped with a stretching mechanism. When the front end of the rolled piece is pulled, the mean rolling pressure is reduced. The control system for the electric motor driving the working rolls of the mill for rolling sheets of varying cross section must be suitable for reversing the mill as well as for operations at several different speeds, thus ensuring that the speed of the rolls is uniform over the whole range of the load. If a preset load is exceeded, a cutoff should operate. These requirements are best met by employing circuits with a dynamoelectric amplifier.

The capacity of the electric motors of the screw-down mechanisms of mills for rolling sheets of varying cross section is considerably higher than that of mills for rolling sheets of uniform thickness since the speed of the displacement of the upper roll of cold and hot-rolling thin-sheet mills for uniform thickness sheets is 10-20 times lower than for mills for rolling sheets of varying cross section. In addition, the length of time

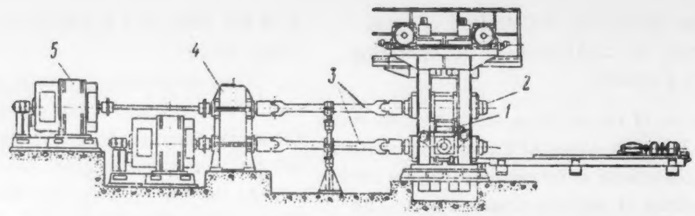


Fig. 4. Four-high reversing mill with driving back-up rolls: 1) working rolls; 2) back-up rolls; 3) spindles; 4) gears; 5) electric motor of the mill.

during which the electric motor of the screw-down mechanism of the thin-sheet hot-rolling mills is switched on constitutes 15-25% while at the mills for rolling sheets of varying cross section it constitutes 60-100% because the rolls are moved during the rolling process as well as during the interval (when the roll is returned to the initial position).

STRIP-ROLLING MILLS

Side by side with an extensive development of the production of wide cold-rolled coiled steel strip, the production of thin strip, 50-400 mm wide and 0.2 mm or even 0.001 mm thick, has increased considerably in recent years.

It is impossible to obtain very thin strip to the required accuracy on the wide-strip mills and therefore it is necessary to develop new mills which differ radically from the old mills and which ensure a higher throughput of the mill and accurate dimensions of the rolled strip.

New reversing and continuous four-stand mills with a new driving system – using the back-up rolls – for the manufacture of low and medium-carbon steel strips of types which are mass-produced have been developed and are being put into operation. On these mills one can use working rolls of very small diameter (100, 150 and 200 mm) which make it possible to produce strip with a high range of initial and final thickness. In addition, the use of back-up rolls as driving rolls (Fig. 4) has a number of advantages: the necks of the back-up rolls do not limit the maximum torque; the rolls can be changed quickly since there is no need to disengage the rolls and spindles; there is no need to install twin rolls of the same diameter since the speed of rolling is determined by the peripheral speed of the back-up rolls. The mill can be used for producing 3.5-ton coils of 300 mm width and 0.1 mm minimum thickness strip at a speed of 10 m/sec.

Reversing mills with working rolls of 40, 55 and 70 mm diameter, driven separately via back-up rolls, shall also be used for rolling 50 mm wide and 0.03 mm thick strip from high-carbon steel. The maximum speed of rolling will be 10 m/sec.

Twenty-roll mills (Fig. 5) with a 7.5 mm/sec speed of rolling will be installed for the production of strip from stainless and electrotechnical steels. The minimum thickness of the strip will be 0.1 mm and the weight of the coil 15 tons. Strips of 300 mm width and 0.02 mm thickness from high-carbon steel and precision alloys will be rolled on 20-roll mills with working rolls of 16-20 mm diameter; the maximum speed of rolling on these mills will be 7.5 m/sec and the weight of coils will be 2 tons. Special 20-roll mills will also be installed for rolling 30 mm wide strip from high-melting alloys with special physical properties.

It has been planned to introduce the automation of the main technological operations, of the inspection and

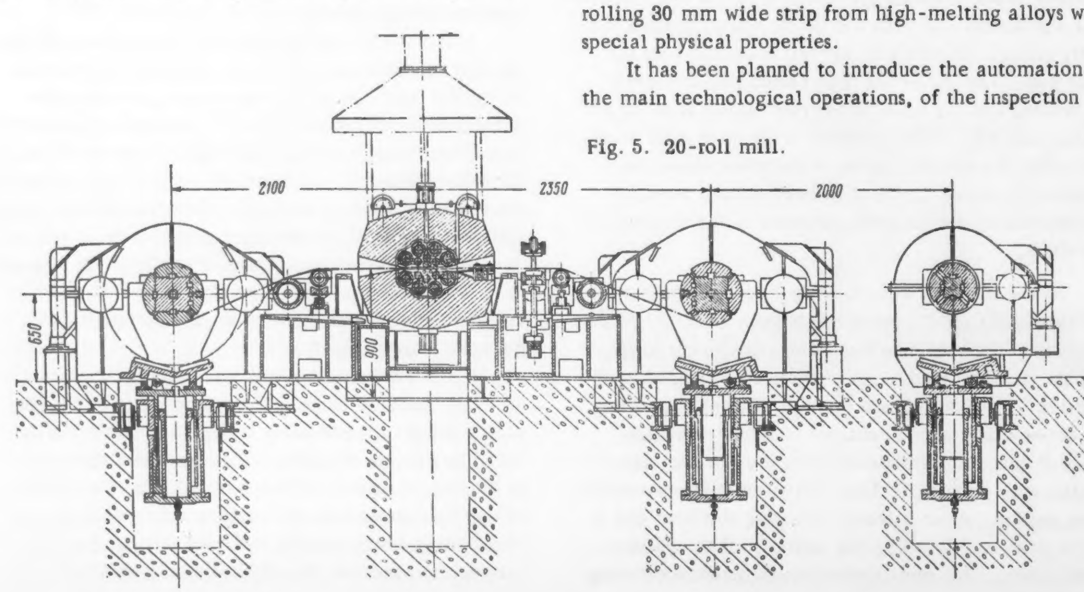


Fig. 5. 20-roll mill.

adjustment of the strip thickness on all rolling mills. A high throughput capacity of the mills will be ensured by employing coils of large weight.

The 20-roll mills make it possible to roll stainless and carbon steels at 50-60% reduction in one pass and at 80-90% total reduction between annealings. The weight of the equipment of the multi-roll mills is reduced by 30-40% compared with ordinary four-high mills; substantial savings in electric power during the annealing of the steel is attained, in particular in the thermal treatment of stainless steel, because one or two annealings are eliminated at each processing stage and this constitutes a considerable saving.

INTER-WORKS COURSE ON TUBE-ROLLING EQUIPMENT

G. I. Shandrenko

VNIIOChERMET

Between November, 1958, and January, 1959, an Inter-Works Course for the study discussion and adoption of advanced techniques in the production of rolling and drawing equipment at the tube rolling mills was held. The course was attended by heads of industrial and metallographical laboratories, group leaders and engineers from TsZL,* equipment foremen and team leaders, supervisors of technical groups, technologists and deputy heads of casting shops, senior roll operators and foremen of tube-rolling shops, shift heads of drawing shops and scientific workers from the UkrNITI and VNIIOChERMET Institutes.

The course was carried out at the Pervoural'sk Novotrubnoi Works, the Sinarsk Works, the Azerbaidzhan Works, the Zakavkaz Works, Nikopol' Yuzhnotrubnoi (YuTZ) Works and the Dnepropetrovsk Works.

At each of these works the members of the course were given lectures by the technical staff on the technology of manufacturing equipment, on its use and on the research carried out by the laboratories with a view to improving the quality of the equipment. The members of the course acquainted themselves directly with the technology of equipment production, operating conditions, durability, maintenance, the methods of acceptance and handing out of equipment and the calculation of the service life; they revealed and discussed shortcomings in the organization and technology of production and the use of equipment; they worked out methods and suggestions for improving the technology of production and the use of equipment.

As a result of the dissemination of the experience gathered during the course, 177 suggestions were put forward. The course considers the adoption of water-cooled nondetachable mandrels on the automatic piercing mills as one of the most important suggestions. By this means the throughput capacity of the mills will be increased, the working conditions of mill operators will be improved and the consumption of mandrels will be substantially reduced. The Lenin Dnepropetrovsk Works was the first to introduce the nondetachable mandrel, and the Azerbaidzan and the Zakavkaz Works have also made good progress in the adoption of the new type of mandrel.

After studying and discussing the experience of all the Works, the course made the following recommendations: to adopt water-cooled mandrels at all piercing mills and to make thick-walled mandrels by forging according to the experience of the Lenin Works; to carry out comparative tests on mandrels made of 12KhN3A and 20N4FA steels; to test conical mountings of water-cooled mandrels and to install an additional cooling for nondetachable mandrels at the place of the initial deformation in accordance with the technique of the Lenin and the Yuzhnotrubnoi Works.

It was suggested that the DIP-200 lathe should be equipped with a hydraulic profiling carriage for the external machining of water-cooled mandrels at the Lenin Works.

The course considered as an equally important measure the mechanization and systematization of separate operations of the technological process of mandrel and guide manufacture for all automatic machines. In this way, the service life of mandrels and guides will be increased, their consumption will be reduced and the output of mills will be increased. At the Novotrubnoi Works, it is necessary to improve the surface quality of cast mandrels of piercing and automatic machines by selecting appropriate mold materials using the experience of the YuTZ; the guides of piercing mills of a large size should be cast into dry molds; the guide molds should be provided with coolers during the casting of the guides and the quality of the metallization of the piercing mandrels should be improved by increasing the surface area and the thickness of the metallized layer and by rotating the mandrel during the metallization process as is being done at the YuTZ.

At the Sinarsk Works, the technology of the production of the mandrels and guides of piercing and automatic mills for the Chelyabinsk Works should be made more effective by improving the gate system in accordance with the experience of the NTZ and YuTZ; the return of all worn tube-piercing equipment from the Chelyabinsk Works should be arranged.

At the Azerbaidzhan Works, the course suggested that the technological instructions for the production of tube-rolling equipment should be amplified more accurately; that the quality of the surface of piercing-mill mandrels should be improved by means of changing to a metallic pattern, improving the strength of mold mixtures, blasting the mold before the assembly, feeding the steel through the nozzle from below, and providing a better lighting of the working place; that machine preparation of the molds for tube rolling equipment should be adopted; that the car-bottom furnace, which is available at the shop, should be used for the thermal treatment of the equipment; that the necessary changes in the technological instructions on the thermal treatment of mandrels for the automatic mill should be introduced;

* Central Works Laboratories.

that normalization and tempering separately or with the use of compressed air blowing in the furnace up to a maximum temperature of 600° C should be carried out; that the quality of the thermal treatment of mandrels should be checked not by the appearance of the oxide film on the surface but by the hardness (Rockwell C 24-40); that the temperature of heating the mandrels of the piercing mill during the thermal treatment should be increased to 970-990° C in accordance with the technique of the YuTZ, the NTZ, and Lenin and other Works.

At the Zakavkaz Works, it is necessary to work out technological charts for the production of all types of equipment on the basis of the experience of the YuTZ; the charge for the electric furnaces and cupola furnaces should be prepared from material of dimensions specified by GOST; the waste material of the equipment (gates scrap, rejects) should be utilized in the charge; each batch of the mold mixture should be released only after satisfactory physical and mechanical properties have been attained; the mold mixtures should be kept in the settling bins for not less than 4 hr the mold materials should be loosened before they are used. In addition, a large number of other suggestions have been adopted.

At the Yuzhnotrubnoi Works, the Course recommended the discontinuation of the use of worn and rejected mandrels for melting cast iron in cupola furnaces; the supply of suitable size scrap of carbon steel and baled tube waste and scrap for the casting shop; the mechanization of the charging of the cupola furnace (the installation of skip hoists); the equipping of the casting shop with more accurate instruments for temperature measurements, etc.

The Course recommended the introduction of improved shapes of mandrels and guides, the use of more durable material and a reduction in the number of types of equipment. For this purpose it is necessary to change the chemical composition of the piercing-mill mandrels; to remove the lower limit of the carbon content, to reduce the upper limit to 0.25%, and to extend the allowable limits of sulfur and phosphorus content to 0.06%; to eliminate nickel from the chemical composition of steels used for automatic-mill mandrels; to change the construction of the guide holder and to introduce the guide for the piercing mill without the flange.

In addition to these general measures, the Course worked out concrete recommendations for individual Works, in particular with regard to increasing the durability of rolls of tube rolling mills and of drawing equipment, improving the quality of tubes as well as improving working conditions.

The Course also made a number of recommendations for scientific research institutes.

THE MANUFACTURE OF OVAL WIRE FROM STEEL 25

Cand. Tech. Sci. Ya. Kh. Sartan

"Proletarskii Trud" Works

According to technical specifications TUM 328-57, finished oval wire should be 4.0 mm thick along the major axis and 2.5 mm along the minor axis with tolerances of ± 0.08 mm; the ultimate tensile strength, σ_B, should be not less than 60 kg/mm^2 and the wires should stand up to at least three bendings.

It is essential to obtain as uniform a reduction of the separate parts of the cross section as possible, in particular in the finishing die. Therefore, the following transitions from circular to oval sections (see figure) were selected; from 6.0 mm diameter circular section to 5.5 x 5.0 mm oval section; from 5.5 oval to 5.0 x 3.8 mm oval; from 5.0 x 3.8 oval to 4.5 x 3.0 mm oval; and from 4.5 x 3.0 to 4.0 x 2.5 mm oval.

Tools of two types were used for making shaped dies: one of the conical shape with oval cross section and another of uniform thickness with oval cross section. The first tool is used for forming the die hole and the other for finishing the bearing zone of the die hole. Usually, the uniform-thickness tool is made of wire of the same cross section.

Low-carbon wire of 6.4 mm diameter was used as the initial material for the preparation of conical tool. The preparation of the forming tool consists of machining a rectangular tapering shape and then filing (with a fine file) two planes to the required dimensions.

For an easier assembly of the die hole, pobedit blanks from VK6 alloy were obtained in a semisintered condition and were shaped to the corresponding contour with an allowance for contraction during the subsequent sintering and an allowance for grinding and polishing.

The technological process of making oval wire in accordance with the accepted technical specifications is as follows:

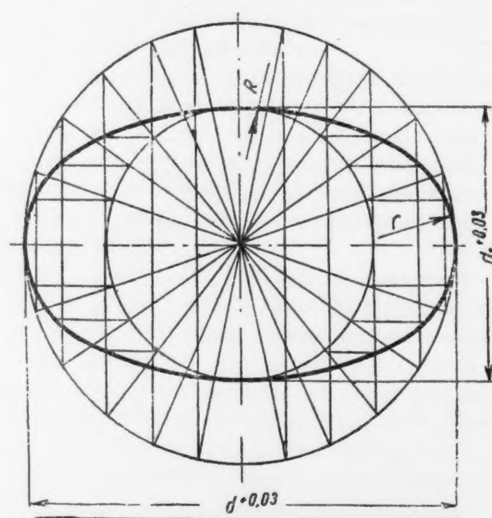
1. Pickling 6.0 mm diameter wire rod, rinsing and lime treating.
2. Drawing the initial 5.5 x 5.0 mm oval section (preliminary oval section) from the wire rod of 6.0 mm diameter; 23.5% reduction.
3. High-temperature tempering of 5.5 x 5.0 mm wire rod.
4. Pickling, rinsing and lime treating.
5. Drawing 5.5 x 3.8 mm oval from 5.5 x 5.0 mm oval section; 31% reduction.
6. Drawing 4.5 x 3.0 mm prefinishing oval from 5.0 x 3.8 mm oval section; 29% reduction.
7. Pickling, rinsing and chemical treatment.
8. Drawing the final 4.5 x 3.0 mm oval section from the 4.0 x 2.5 mm section; 28% reduction.
9. Tests and acceptance by the OTK.
10. Delivery to storage.

The wire rod and intermediate sections are pickled in 18-8% sulfuric acid solution for 10-20 min; rinsing and lime treating is carried out in accordance with the current instructions. The chemical treatment is carried out after the pickling and rinsing. This treatment consists of exposing rinsed wire for 45-60 min in air so that the surface of the metal is covered with a yellow layer of iron oxides.

The high-temperature tempering is carried out in electric furnaces of ShO-130 type at 680°C for 1 hr with subsequent cooling in soaking pits. The wire rod and intermediate sections are drawn on a single stage drawing machine at a speed of 100 m/min with the use of a dry lubricant (soap powder). The drawing proceeds normally and no fractures occur.

Tests on an experimental batch of the wire gave the following results: $\sigma_B = 78$ kg/mm^2 (mean); number of bendings: six around the major axis and ten around the minor axis. These results conform with the technical specifications.

The worked out technological process and the arrangement of passes make it possible to produce 4.5 x 2.5 mm oval wire from steel 25 and fully satisfy the TUM 328-57 technical specifications.



No. of drawing	$d^{-0,03}$	$d_1^{-0,08}$	R	r
1	5,42	4,92	3,05	2,35
2	4,92	3,72	3,10	1,75
3	4,42	2,92	3,00	1,10
4	3,92	2,42	2,80	0,90

The construction of the oval pass by transition from the circular cross section.

M. A. Bukshtein. Steel Ropes.
Moscow, Metallurgizdat, 1959, 150 pages

The book contains a description of a rational design of ropes for various purposes and operating conditions; various steels, wires and cores for rope manufacture are discussed; methods of twisting rope from circular and special shape wires and the type of coatings for steel wire and their design weight are given. Methods of selecting rational working conditions for the rope, selecting the diameters of drums and pulleys, their relative location and selecting the safety factors for ropes are recommended.

Ropes with point contact, line contact, and point and line contact of the wire, as well as shaped-strand, multistrand, flat and spiral ropes of closed construction are described. Conditions for the normal operation of the rope are recommended.

A sample of a book for recording the results of inspection of hoisting ropes and recording the consumption of the ropes is given in the appendix.

The book is intended as a reference text for foremen and skilled workers engaged on operations involving the use of steel rope.

N. N. DOBROKHOTOV

Cand. Tech. Sci. B. Kh. Khan

In March, 1959, the Soviet people marked the 70th birthday of one of our outstanding metallurgists, Nikolai Nikolaevich Dobrokhotov.

Nikolai Nikolaevich Dobrokhotov was born in 1889 to the family of a telegraph operator in Arzamas. He was educated in Nizhii Novgorod where in 1907 he graduated from the secondary school. In the same year, after passing scholarship examinations, Nikolai Nikolaevich Dobrokhotov entered the Mining Plant Department of the Petersburg Mining Institute. During his student years he spent a considerable time at various plants, worked as a designer, as a foreman at an open-hearth shop and held some other posts. In 1914, Nikolai Nikolaevich graduated from the Mining Institute and joined the open-hearth shop of the Perm Ordnance Factory where he was engaged on mastering and improving the technology of artillery steel production. While working in industry, Nikolai Nikolaevich continuously widened his scientific and technical knowledge. In 1921, he joined the Metallurgy Department of the Petersburg Mining Institute as an Assistant Lecturer, and, in 1924 he was appointed to the Chair of Steel Metallurgy and Metallurgical Furnaces at the Sverdlovsk Industrial Institute.

He worked in the Urals until 1931, then he moved to Moscow where he held the post of Head of the Furnace Laboratory of the TsNIITMASh. In 1936, he was invited to take the Chair of Steel Metallurgy and Furnaces at the Dnepropetrovsk Metallurgical Institute. He worked there until 1941. In 1938, N. N. Dobrokhotov obtained the degree of Doctor of Technical Sciences, and, in 1939, he became a full member of the Academy of Sciences of the Ukrainian SSR.

During the Patriotic War, Nikolai Nikolaevich lived in the Urals where he was of great assistance to plants which made artillery and armor steels for war needs. In 1944, he returned to the Ukraine where he became the Head of the Steelmaking Section of the Institute of Ferrous Metallurgy of the Academy of Sciences of the Ukrainian SSR and the Head of the Department of Steel Metallurgy and Furnaces at the Kiev Polytechnical Institute.

In 1949, on his initiative, the Gas Utilization Institute of the Academy of Sciences of the Ukrainian SSR was set up and he became its first director. In the same year he was elected Chairman of the Technical Sciences Section of the Academy of Sciences of the Ukrainian SSR. He remained in this post until 1952. At present, he is the Head of the Department of Gas Utilization in Metallurgy at the Institute which he founded.

The scientific and engineering activities of N. N. Dobrokhotov are extensive, many-sided and closely interlinked with the successes of our metallurgical industry. As a great specialist in the field of steel metallurgy and furnace and gas technology, Nikolai Nikolaevich makes use of his vast knowledge in these fields when solving scientific problems. He has published more than a hundred scientific works on various metallurgical problems.

The first works of N. N. Dobrokhotov on the theory of the gas-producing process and the study of the gasification of Siberian coals constituted the basis of the design of gas producers for the Kuznetsk Metallurgical Combine. The design method developed by N. N. Dobrokhotov for the gas producing process still remains valid.

N. N. Dobrokhotov is one of the founders of the modern theory of design and construction of industrial furnaces. The theory which he developed constitutes the basis of all modern courses on gas-furnace technology. Nikolai Nikolaevich was the first to criticize the "hydraulic" theory of furnaces, put forward by V. E. Grum-Grzhimailo, and proved that his assumptions were correct. The application of Dobrokhotov's "energy theory" of furnaces in practice brought about tremendous advances in gas-furnace technology.

Under N. N. Dobrokhotov, the Department of Steel Metallurgy and Furnaces at the Dnepropetrovsk Metallurgical Institute extended its activity significantly in the prewar years. There the gifts of Nikolai Nikolaevich, his purposefulness and his devotion to technical progress manifested themselves to the full. In accordance with his design, all open hearth furnaces were equipped with forced draft fans, the bath of the furnaces was widened and made deeper, the furnace ports were improved; for the first time in our country an open hearth furnace with charging doors without an arch was built. These and some other measures made it possible to increase the output of open hearth furnace shops by a factor of 1.5-2 without introducing new furnaces.

At the same time, N. N. Dobrokhotov and his students have developed the principles of the modern technology of steelmaking which have been widely accepted at metallurgical works. The basic elements of this technology can be summarized as follows: high-rates of carbon loss during the pure-boil period, intensified thermal and temperature regimes of the melting process, the rejection of the old manganese regime, tapping the heat without a preliminary deoxidation with blast-furnace ferrosilicon, etc. These theoretical considerations, initially debatable, were subsequently confirmed in practice and now one can

say with confidence that the technology of steelmaking in basic open-hearth furnaces as applied at present has developed to a great extent under the influence of the research work and publications of N. N. Dobrokhotov. In 1952, he suggested that the addition of alloys and the deoxidation of steel with hard ferroalloys completely in the ladle should be carried out. This method was successfully tested and adopted at several works and resulted in a substantial reduction in the cost of steel production. The publications of N. N. Dobrokhotov in the field of the technology of steelmaking have been reviewed in a booklet called "Contemporary Technology of Steelmaking in Open Hearth Furnaces."

Nikolai Nikolaevich works incessantly on the theoretical aspects of metallurgical processes. As early as 1928, he established the role of the surface tension of liquid steel in the process of the formation of carbon monoxide bubbles in the boiling bath. The equation which he proposed has been incorporated in all text books on steel metallurgy and the theory of metallurgical processes. Nikolai Nikolaevich was one of the first scientists who introduced the entropy concept into calculations of the steelmaking processes instead of equations containing arbitrary physicochemical constants. He has analyzed the basic reactions of the steelmaking process (deoxidation, degasification, desulfurization and dephosphorization of steel) with the use of the ionic theory of liquid slags. The theoretical works of N. N. Dobrokhotov have been published in a monogram "The Application of Thermodynamics in Metallurgy." His publications in the field of dynamics of diffusional processes, his theory of drying ceramic products, his theory of steel deoxidation, his theory of slag structure, his theory of gas producing and his other works, are well known.

At present, N. N. Dobrokhotov is supervising important research work on a further improvement of steelmaking production, on the conversion of open hearth furnaces to natural gas firing, on the direct production of iron from ores and on the magnitizing roasting of iron ores.

N. N. Dobrokhotov considers an organic link between science and industry and the adoption of new and advanced techniques into industry of great importance and he keeps in close contact with many metallurgical and machine building works.

The Party and the Government appreciated N. N. Dobrokhotov's merits. He has been awarded Lenin's Order, two Orders of the Red Banner of Labor, several medals and the title of Honorable Scientist and Technologist of the Ukrainian SSR.

MAGNETIZING ROASTING OF IRON ORES

Cand. Tech. Sci. S. K. Grebnev

Experience of Czechoslovak Metallurgists

While on a visit to the Czechoslovak Republic, we had an opportunity to acquaint ourselves with the work of research institutes and industrial establishments engaged on ore beneficiation.

The magnetizing-roasting beneficiation shop of the iron and steel works in the town of Trznets processes partially oxidized large-inclusion carbonate iron ores from 9 different mines. The average iron content in ore mixture is about 27%, the iron content in the form of carbonate consitutes 70-90% of the total iron. Nevertheless, during the roasting without the addition of solid reducing agent, these considerable variations in the composition of the ore do not affect the final result. According to the investigations of Czechoslovak scientists for a successful magnetizing-roasting of ores in which the hematite is the main iron-containing constituent, it is necessary to add 2% (by weight of ore) of pulverized brown or ordinary coal to the ore.

The ratio of (CaO + MgO) to SiO2 in the ore is 0.5; the moisture content of the ore is 7%. The particle size of the ore for roasting is below 25 mm; the <3mm fraction constitutes about 80%.

The shop contains four roasting units (Fig.) each of which consists of a 47.5 x 2.4 m (181 m^3 working volume) roasting furnace and a 25.0 x 1.5 m (40 m^3) cooler. The delivery end of the furnace and the charging end of the cooler are located in a stationary chamber with a guiding trough on which the roasted ore from the furnace passes to the cooler. At the same time, the ore is sprayed with water so that the cooling process is speeded up and the steam produced establishes a positive pressure in the chamber, thus preventing the leakage of air into the chamber. The joint between the chamber furnace and the cooler is made tight with the use of spring rings. Air is not allowed to enter the chamber because it would oxidize the ore and affect its magnetic properties. The steam is drawn off into a separate tube which is connected to the delivery end of the cooler; in this way, the ore is cooled in a steam atmosphere.

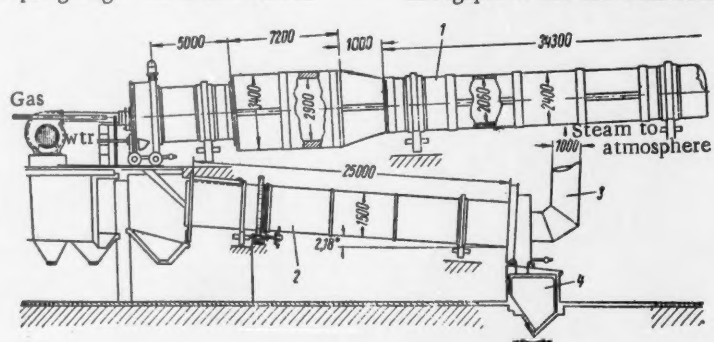
The temperature of roasting is 800-850° C; in the furnace itself, the ore cools down to 700° C and in the cooler to 100° C. The temperature of waste gases is 250° C.

The furnace is fired by means of an end burner with a mixture of blast-furnace, coke-oven and natural gases of calorific value approximately 3300 kcal/m^3. The use of gas of a lower calorific value causes a rapid fall in the throughput capacity of the furnace, and on the other hand, with a higher calorific value of the gas the ore becomes overheated and the roasting process is upset. It is recommended to use gas of a higher calorific value outside the furnace and allow only the combustion products into the furnace.

At the metallurgical combine in Rudniany, tailings of 5000 kcal/kg calorific value from coal washing plants are used as fuel for furnaces for the magnetizing roasting of the same ores, and now it is intended to change over to fuel oil which was used previously. The technological results and heat consumption in the roasting process are the same for all three types of fuel (gas, fuel oil and tailings).

The furnaces for the magnetizing roasting have forced draft, and the cooling drums have natural draft. Each furnace is equipped with an exhaust fan of 24,000 m^3/hr capacity. The cross section of the tube for steam draw-off from the cooler is 2.2 times smaller than the cross section of the cooler.

With the above operating conditions the raw ore throughput of one unit constitutes 700-750 ton/24 hrs



Plant for the magnetizing roasting of ores: 1) furnace; 2) cooler; 3) tube for drawing off steam; 4) bin for cooled ore.

which corresponds to an output of 4 tons per 24 hours per one m^3 of furnace volume. The fuel consumption in terms of conventional fuel (700 kcal/kg) constitutes 4.2% by weight of raw ore.

Steps have been taken with view to reducing the fuel consumption. The personnel at the factory is confident that after the reorganization of the fuel system the consumption of fuel will be reduced to 3.5%.

The furnaces are lined with acid firebricks which are laid on a cement-asbestos mortar of 15 mm thickness. Over a length of 6 m from the charging end, the furnace is lined with steel or cast iron plates, $6000 \cdot 333 \cdot 20$ mm, laid on concrete and bolted to the furnace shell. Fins, $300 \cdot 300$ mm, made of steel sheets, are welded to the plates at an angle of 20°. The fins form 3 turns of a helix which assist the movement of materials.

The service life of the steel plates is six months; the fire brick within 30 m of the charging section lasts for three months, and at the charging end it lasts 18 months.

The cooling drum is lined simply with steel plates which are fixed to the shell; their service life is 2 years. The plates which are used for lining the furnace and the cooler should be flat since shaped plates have a lower resistance to wear. The actual operating time of the roasting furnaces does not exceed 75% of the total time; the main idle periods are caused by unscheduled changes of the furnace lining and other mechanical defects. It can be assumed that with an appropriate planning of preventive repairs, the actual operating time of the furnace could be increased to 85-90% of the total time.

After the roasting, the ore, which contains 35-37% iron, is crushed down to 3 mm particle size and beneficiated on ordinary rotary magnetic separators. The concentrate contains 49-52% iron, the yield is 64% and the extraction 80%, the basicity of the concentrate is greater than 1 and this enhances its value as a raw material for smelting.

The technique of the Czechoslovak plant may be utilized in the USSR. One should use the same type of machines for roasting, but because the Soviet ores differ from the Czechoslovak ores we consider that it is essential to add up to 2% brown coal or ordinary coal fines to the ore before the roasting; and one should let the waste gases into the transfer section after they have been freed from dust.

The temperature of roasting should in our opinion, be 800-850°C, and judging by practical experience, the temperature of the waste gases should be 150-350°C. The temperature should be raised if during the roasting it is necessary to remove various harmful impurities (for instance arsenic from Kerch ores) from the ore.

It is advantageous to use natural gas — the cheapest fuel — but, depending on local conditions, one can also employ fuel oil as well as cheap brown coal or tailing from coal washing plants.

The gases leaving the roasting furnaces contain a large amount of dust and they should be cleaned with the use of units similar to those which are in operation at all new cement plants and which remove up to 99.5% dust. If the gases also contain other harmful substances e.g., arsenic, when Kerch ores are roasted it is necessary to provide equipment for continuous dust removal as this will improve working conditions and show some savings. The process schemes for the magnetic beneficiation of each ore should be worked out with taking into account the specific properties of the ores concerned.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us – Publisher.